MINING engineering

NOVEMBER 1954

ARCTIC DREDGING

WILFLEY PUMPS

Reduce Pumping Costs...Increase Production

Minimum downtime required for replacement of parts

Continuous operation without attention

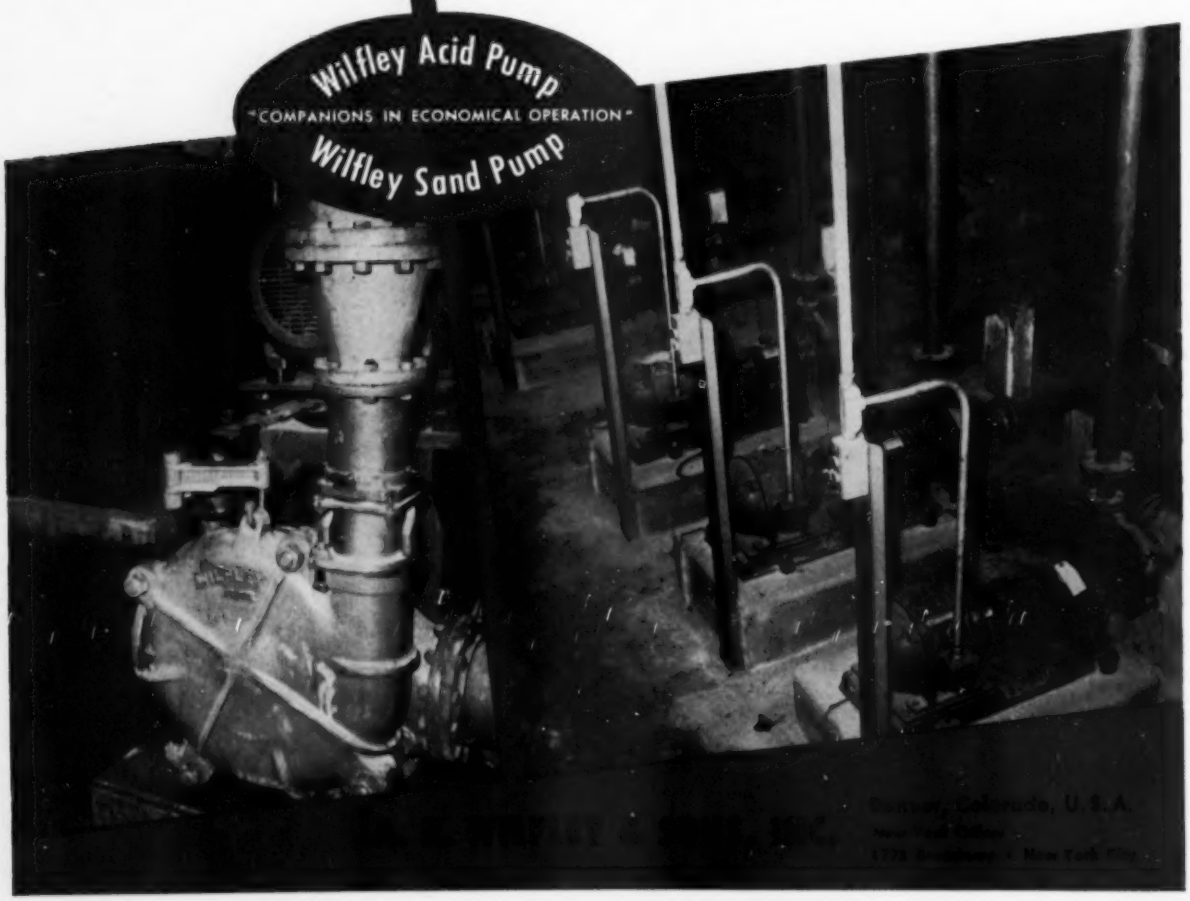
Maintained High Efficiency

Economical pump for every requirement

Individual engineering on every application. Write, wire or phone for complete details.

Wilfley's remarkable success-record in creating substantial dollar savings in power and maintenance means low-cost production on every pumping job. This cost-saving efficiency is born of engineering "know-how" plus many years of experience in keeping pace with the changing requirements of modern industry.

Shown here are typical Wilfley installations. Rubber lining, alloy irons, plastics and stainless steel are some of the materials used in Wilfley Sand Pumps and Acid Pumps to provide maximum pumping efficiency.



MINING engineering

VOL. 6 NO. 11 NOVEMBER 1954

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COVER

Artist Herb McClure finds dramatic accent this month in the power of the dredge bucket line. But even the bucket line is brought to a halt by Arctic winters. Patrick O'Neill's article on page 1068 tells some of this story—the battle against ice and frost to extend the dredging season.

FEATURES

Personnel Service	1038	Drift	1067
Manufacturers News	1044	AIME News	1117
Books	1045	Personals	1125
Reporter	1049	Professional Services	1130
Mining News	1051	Coming Events	1132
Trends	1064	Advertisers' Index	1132

ARTICLES

Arctic Gold Dredging — Patrick H. O'Neill — 1068

Vermont Copper Uses Mucking Machines for Stope Drawing — Clinton L. Miller — 1072

Development of Monsanto's Western Phosphate Operation — G. Donald Emigh — 1077

Rock Bolting Finds Wide Application — Edward Thomas — 1080

Longhole Drilling for Pillar Extraction — Harry Bapty — 1086

Facts and Figures on the Atlas Shaft Sinking Project — William H. Love — 1090

TRANSACTIONS

Development and Construction of a Ground-Water Supply — Owen F. Jensen — 1093

Structure and Mineralization at Silver Bell, Ariz. — Kenyon E. Richard and James H. Courtright — 1095

Discussion — 1100

— Personnel Service —

THE following employment items are made available to AIME members on a non-profit basis by the Engineering Societies Personnel Service Inc., operating in cooperation with the Four Founder Societies. Local offices of the Personnel Service are at 8 W. 40th St., **New York 18;** 100 Farnsworth Ave., **Detroit;** 57 Post St., **San Francisco;** 84 E. Randolph St., **Chicago 1.** Applicants should address all mail to the proper key numbers in care of the New York office and include 6c in stamps for forwarding and returning application. The applicant agrees, if placed in a position by means of the Service, to pay the placement fee listed by the Service. AIME members may secure a weekly bulletin of positions available for $3.50 a quarter, $12 a year.

MEN AVAILABLE

Mining Engineer, 29, married, 2 children, B.S. mining engineering, University of Washington, 1951. Nine months summer employment underground, 17 months junior engineer, and 3 months experience shift boss underground trackless operation. Veteran World War II and 18 months in Aviation Engineers. Available immediately. Prefers West. M-102-906-E-San Francisco.

Mining Engineer, graduate; 34, married; 5 years experience in engineering and mining—stopes, block caving, also some open pit and construction. Desires position in engineering, mine planning, and design with reputable company. Now employed in New York vicinity. M-105.

Mill Superintendent—Metallurgist offers integrity, initiative, ingenuity. Fifteen years varied experience. Age 36, married, one child. Available short notice. M-103-991-Chicago.

Mining Engineer, 33, married, 3 children, graduate University of Pittsburgh. Holds Pennsylvania First Grade Mine Foreman Certificate. Ten years experience all types mechanical coal mining. Duties include time study, cost analysis and control, determining standard procedures, and mine supervision. Desires responsible position and advancement opportunity for willing ambitious worker. M-104-Detroit.

Metallurgical Engineer, Dutch, equivalent M.S., 30, family. Now employed South America. Three years experience test and operational work large U.S. company. Speaks and writes English, French, Spanish, German. Desires job in mill; will go anywhere. M-130.

POSITIONS OPEN

Geologist with exploration and development experience covering sulphur dome deposits. Salary open. Location, Mexico. F500.

Mining Geophysicists, 2, to take part in pioneer geophysical exploration and development program. Good background in physics and electronics essential. Location, West. W290.

Assistant Mill Superintendent, young, with experience in ore dressing for lithium processing. Salary, $6000 to $7000 a year. W520(a).

(Continued on page 1039)

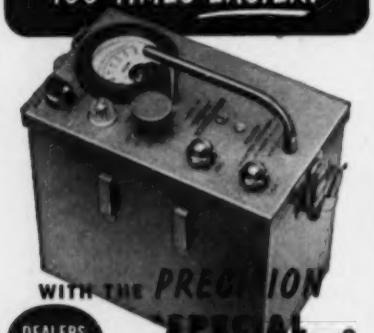
Wanted: first-class firms in U.S.A. for sale of special range of modern machinery screening means, conveyor belts and other specialties for the mining industry. For full particulars, address:

N. V. TEMA, THE HAGUE (Holland), Nassaulaan 1.

AVAILABLE

PLANT DESIGN ENGINEER—Age 36, married, available immediately. Experienced in design of aluminum reduction, gravity separation, magnetic, and flotation concentration plants in supervisory capacity. Especially interested in organization and industrial relations. Relocate anywhere in U.S. or Canada. Box 106, AIME, 29 W. 39th St., New York 18, N. Y.

DETECT URANIUM DEPOSITS 100 TIMES EASIER!



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Here is the ideal field instrument for Uranium prospecting. It's ultra-sensitive, low in cost, lightweight and small in size.

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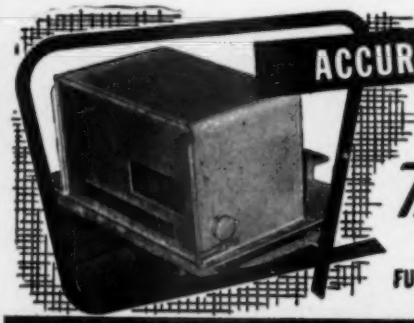

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FULLY AUTOMATIC • EASILY INSTALLED
Write for Bulletin 301
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Splits Feed Any Way You Require

When it comes to splitting pulped feed between concentrating tables or other machines in battery, equally or any other way you wish, the CONCENCO Revolving Feed Distributor is practically foolproof. One Distributor will feed up to ten or more circuits accurately and efficiently. It is sturdily constructed and is equipped with a motor to fit your current characteristics. Power consumption is low. Send for full information and engineering data.

THE DEISTER CONCENTRATOR COMPANY
The Original Deister Co., Incorporated 1906
923 Glasgow Ave. Fort Wayne, Ind. U.S.A.

POSITIONS OPEN

(CONTINUED)

Safety & Ventilation Engineer, minimum age 35; experience in underground metal mining essential. Capable of making mine ventilation surveys and promoting mine safety program. Salary, $5500 to $6000 a year. Permanent. Location, Northwest. S-193.

Process Engineer, crushing. M.E. or E.M. Mining and quarrying background, and knowledge of processing and handling equipment. Duties will be processing, large rock quarrying operations, and beneficiation problems on crushing and grinding for a manufacturer of building materials. Salary, $7200 to $8400 a year. Location, Chicago. C2271(h).

Chief Engineer, 40 to 50, thoroughly experienced in the processing of nonmetallic ore; should have at least 20 years experience in engineering, including design, plant layout, etc. Must be familiar with mining equipment, flotation, jigs, ball mills, centrifuges, conveyor systems, etc. Some travel. Salary about $10,000 a year. Location, Southwest. W287.

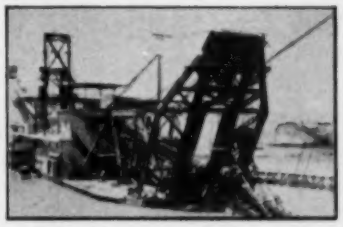
Mining Engineer, graduate, at least 2 years experience. Duties will include handling of contract development, mapping, compilation of progress reports, measuring stopes, logging drill cores, and performing other related engineering work. Location, South. W225.

Production Shift Engineer, graduate metallurgical or chemical engineer to take complete charge of shift of lead smelter. Salary, $5400 to $6000 a year. Location, Midwest. W299(b).

Metallurgical Engineer, to take charge of the metallurgical problems of a mining operation. Should have a thorough knowledge of flotation, cyanidation, lixiviation, and if possible, some knowledge of running a sulphuric acid plant. Accommodations for man and wife only. Salary open. Location, Chile. F172.

MINE SUPERINTENDENT

Exceptional opportunity for very hard-working mining engineer with strong interest in geology, 30 to 40 yr. old, with demonstrated supervisory ability and experience particularly underground mining methods for work at mine operated by chemical company to insure mercury supply. Write fully, including photograph, resume of college records, experience and references to Dr. S. J. Buckman, Buckman Mines, Geyser Road, Cloverdale, Calif.



M-8 jig developed by YUBA for concentrating ores on dredges and in mills. It's designed to save space, reduce downtime, increase production.

FULL AREA OF

YUBA JIG BED

IS COMPLETELY ACTIVATED

YUBA jig action is positive. You set the speed and stroke wanted, get constant, even pulsations that create surface action over full area of bed. Result: YUBA M-8 jigs have a large material capacity per flow line under full control.

Any material that can be concentrated can be handled successfully in YUBA jigs, including . . .

CASSITERITE	RUTILE
GOLD	SCHEELITE
PLATINUM, ETC.	GARNETS
MONAZITE	SAPPHIRES
ZIRCONIUM	IRON
ILMENITE	COPPER

Design Eliminates Trouble Spots

Stainless steel hutch valves and screens prevent rusting and clogging. Rubber seal between screen grids and basket confines action to screen area. Long-wearing hutch diaphragms of reinforced synthetic rubber can be replaced easily.

"Package Drive" units for YUBA jigs are interchangeable, completely enclosed, self-lubricating. Generous use of antifriction bearings reduces power required. Maximum frequency of a 4-cell M-8 jig is 350 at ¼". Stroke adjustments between ¼" minimum and 3" maximum are easily and quickly made, enabling you to closely control jig action.

YUBA jigs can be installed in new or old dredges or mills to supplement existing jigs or to replace other concentration methods. Send us data on ore, feed sizes and present installation if you wish us to furnish details to adapt YUBA jigs to your operation.

71

YUBA DREDGES — PROVED DEPENDABLE

YUBA MANUFACTURING CO.

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ABRASIVES *have been the* "Silent Partners" *of World Progress*

Since ancient man's first need for tools and weapons, abrasives have been a decisive factor in shaping the progress of civilization. As man's manufacturing abilities increased, natural abrasives were no longer sufficient . . . which led to the use of the first manufactured grinding wheel. Today, practically every man-made product depends upon abrasives.

Of utmost importance in processing, prior to the manufacture of the finished abrasive product, is the method of reducing the hard component materials into workable sizes as quickly and efficiently as possible. *Here, as in all other fields where fast, low cost reduction is paramount for profitable production, SYMONS* Cone Crushers are widely used by the leading producers of abrasive products . . . to produce a constant, uniform size of material.*

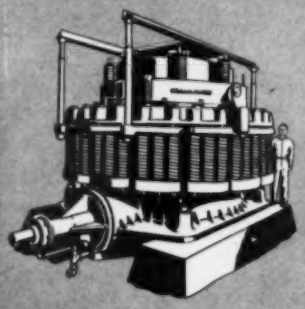
NORDBERG MFG. CO., Milwaukee, Wisconsin C-253

NORDBERG

MACHINERY FOR PROCESSING ORES and INDUSTRIAL MINERALS

NEW YORK • SAN FRANCISCO • DULUTH • WASHINGTON • TORONTO
MEXICO, D.F. • LONDON • PARIS • JOHANNESBURG

Statue of "The Grinder", represents a Scythian slave whetting a knife on an irregular shaped, natural sharpening stone.



SYMONS* Cone Crushers . . . the machines that revolutionized crushing practice . . . are built in Standard, Short Head, and Intermediate types, with crushing heads from 22 inches to 7 feet in diameter — in capacities from 6 to 900 tons per hour.

*SYMONS . . . A REGISTERED NORDBERG TRADEMARK KNOWN THROUGHOUT THE WORLD.

SYMONS PRIMARY CRUSHERS

GRINDING MILLS

MINE HOISTS

SYMONS VIBRATING GRIZZLIES and SCREENS

DIESEL ENGINES

BEST...

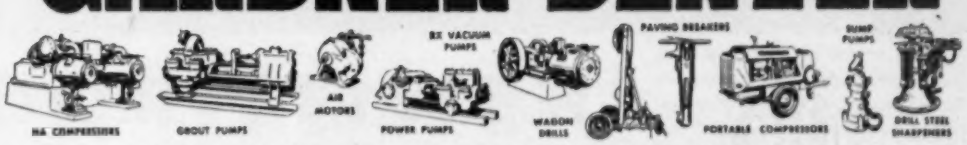
for any stoping job...

a GARDNER-DENVER RB

- Two sizes: 2 ¾" and 3 ⅛".
- Self-Rotating.
- Tappet or Collared Chuck.
- Rotary or Push-Button Feed Leg Control.
- Direct or Telescopic-Direct Feed Leg.
- *Exclusive* Automatic Water Control Valve if Desired.

SINCE 1859

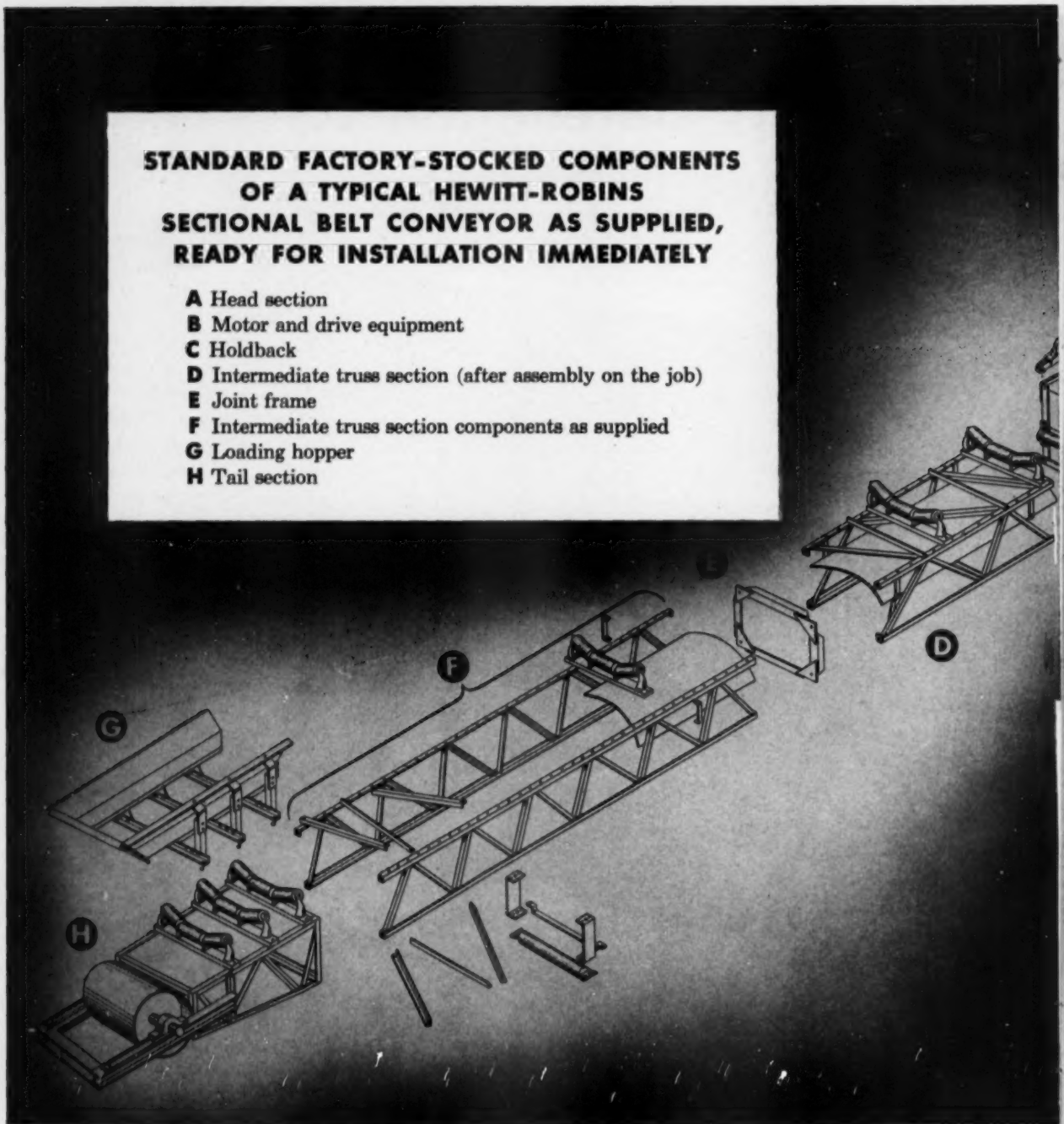
GARDNER-DENVER



THE QUALITY LEADER IN COMPRESSORS, PUMPS AND ROCK DRILLS FOR CONSTRUCTION, MINING, PETROLEUM AND GENERAL INDUSTRY

Gardner-Denver Company, Quincy, Illinois • In Canada: Gardner-Denver Company (Canada), Ltd., 14 Curity Avenue, Toronto 16, Ontario

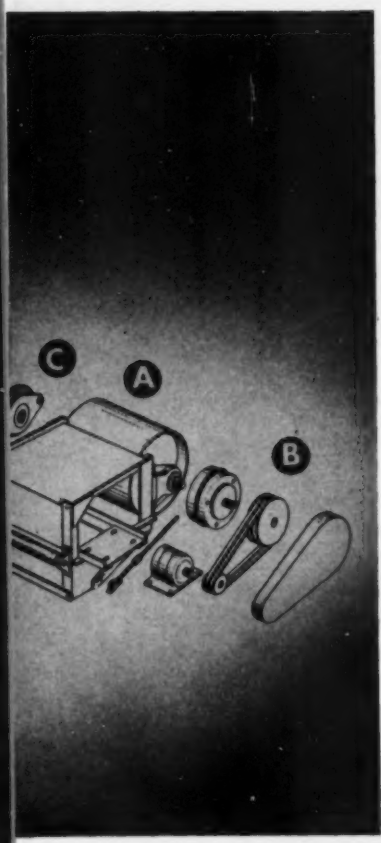
Order Your Belt Conveyor



HEWITT-ROBINS

EXECUTIVE OFFICES, STAMFORD, CONNECTICUT

Right From Factory Stock



Save Time and Money . . . Install your own Hewitt-Robins Sectional Belt Conveyor from Easy-to-assemble Factory-built Components

Now, for the convenience of large and small plant operators alike—Hewitt-Robins offers a standardized sectional conveyor adaptable to a wide variety of uses.

Hewitt-Robins Sectional Conveyors consist of completely assembled factory-stocked components that can be quickly and economically installed and maintained by your own personnel.

When you order a Hewitt-Robins Sectional Conveyor you can be sure of prompt delivery . . . all components and parts from machinery to belting are shipped to you direct from standard factory stocks . . . saves you time and saves you money.

For prompt information about the Sectional Conveyor and any other Hewitt-Robins bulk material handling products, contact the Hewitt-Robins office nearest you or write direct to Passaic, New Jersey.

ENGINEERING DATA

APPLICATION: Designed to handle bulk and packaged materials where special limitations and conditions do not require the use of custom-engineered belt conveyors.

MATERIALS HANDLED: Sand, gravel, run-of-mine ore, coke, stone, coal, gypsum, grain, sugar beets and any other bulk materials that can be handled more efficiently and economically by belt conveyor.

CAPACITIES: From 20 to 500 TPH at belt speeds from 200 to 450 feet per minute.

STANDARD WIDTHS: 18", 24", 30", 36".

LENGTH: From 15'6" up to any length consistent with good conveyor practice . . . inclines up to 22½°.

STANDARD HORSEPOWER: From 2 to 30 hp.

IDLERS: 4" or 5" diameter troughing and return idlers available. Both types have exclusive Hewitt-Robins patented Triple Grease Seal and one-shot lubrication.

BELTING: Complete selection of all Hewitt-Robins standard stocked belts. Special belts available when necessary.

INSTALLATION: All standard factory-assembled components are clearly labeled and shipped with complete instructions, for quick, easy assembly by your own personnel.

Write for Bulletin #132-B on your company letterhead

Here is a partial list of
HEWITT-ROBINS PRODUCTS
that will help you cut handling costs and increase operating efficiency

MACHINERY:

Belt Conveyors
Belt & Bucket Elevators
Car Shakeouts
Screen Cloth
Vibrating Conveyors & Screens

INDUSTRIAL RUBBER PRODUCTS:

BELTING:
Conveyor
Elevator
Transmission

HOSE:
Hewitt-Robins makes over 1,000 different types of industrial rubber hose, including Twin-Weld®, the patented twin-line welding hose.

For information and service on industrial rubber products, contact your Hewitt-Robins Industrial Supply Distributor. Through his complete stock of Hewitt-Robins Rubber Products, and his familiarity with local field conditions, he can fill your supply needs promptly and correctly. See Classified Phone Book for the Hewitt-Robins Industrial Supply Distributor serving your area.

3081

INCORPORATED

DOMESTIC DIVISIONS: Hewitt Rubber • Robins Conveyors • Robins Engineers • Restfoam
FOREIGN SUBSIDIARIES: Hewitt-Robins (Canada) Ltd., Montreal • Hewitt-Robins Internationale, Paris, France • Robins Conveyors (S. A.) Ltd., Johannesburg • **EXPORT DEPARTMENT:** New York City.

Manufacturers News

New Products • *FILL OUT THE POSTCARD FOR MORE INFORMATION* • Equipment

Rotary Drills

Tenth *Bucyrus-Erie* model 50-R rotary drill has gone into service with Calaveras Cement Co. Hitherto used in coal and iron ore operations, the Calaveras installation marks first cement quarry use of the 54-ton machine, largest industrial drill built. In a recent paper it was stated that the $115,000 item, on the basis of direct costs alone, paid for itself in 212 days. Capable of 12¼-in. diam holes, 9⅞-in. bits are used in the California cement operation. **Circle No. 1**

Stopping Loaders

Tru-Stop disk type emergency brakes are now being installed on the 34,000-lb *Eimco* 105 tractor-loader. Two of the brakes are used for track and parking purposes, and

a third in the bucket mechanism. The disk brakes are a product of the American Chain & Cable Co. For more data on this and other features of the Eimco 105. **Circle No. 2**

Ventilation

Cast-aluminum alloy blades of *Jeffrey Mfg. Co.'s* 67 Aerodyne fan are universally adjustable. Overhung drive can handle up to 125 hp belt

driven and higher hp with direct drive. Fan delivers up to 5-in. water gage pressure. **Circle No. 3**

Flow Data

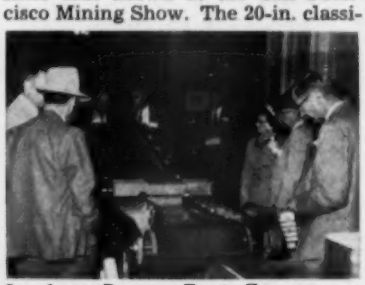
Electromagnetic flowmeter for 2 to 8-in. pipelines, introduced by the *Foxboro Co.*, measures volume flow rate of any liquid of sufficient conductance and velocity. Meter is unaffected by pressure, density, or changes in conductivity. **Circle No. 4**

Sintering

Under a licensing arrangement with Lurgi Chemie of Germany, the machinery div. of *Dravo Corp.* has added iron ore sintering and pelletizing equipment to its line of machinery. Dravo-Lurgi sintering machinery has the background of Lurgi's 40-year experience as the oldest European designer of sintering machinery, plus Dravo's 60 years of manufacturing and engineering experience. **Circle No. 5**

Denver-Finney Classifier

Photo shows visitors crowding around the new Denver-Finney classifier first shown at the San Francisco Mining Show. The 20-in. classi-



fier from *Denver Eqpt. Co.*, uses an endless rubber conveyor belt to replace conventional mechanical raking. The classifier pool is formed by troughing the belt. **Circle No. 6**

Tractor Jumbo

A completely self-contained and mobile drilling rig from *Gardner-Denver* is being offered mining operators and contractors. Gardner-Denver rock drills, G-D hydraulic booms, and a complete G-D com-

pressor are all mounted on a Caterpillar D-8 tractor. Compressor assemblies are available in capacities of 315 to 600 cfm, for direct connection to the rear power take-off of the D-8. **Circle No. 7**

New Reagent

Flocculating agent to reduce product loss, increase settling rate of dispersed materials, and improve filtration is Separan 2610 just introduced by the *Dow Chemical Co.* Proven to increase filtering rate while decreasing moisture content of filter cake, material is claimed effective in extremely small quantities, and useful over wide pH range. **Circle No. 8**

Cyclones

The Dorr Co. has redesigned its standard 6, 12, and 24-in. type FR Dorrclones to use replaceable

molded rubber inserts. Considerable increase in life of sections not previously lined, easy installation, and lower maintenance costs are claimed. **Circle No. 9**

Reagent Feeder

Latest precision liquid reagent feeder from *The Clarkson Co.* is fabricated entirely from plastic, except for motor and name plate. High tensile strength and resistance to acids,

alkalies, and oils are said to result from construction utilizing B. F. Goodrich's Geon vinyl resin in place of stainless steel used in earlier model. **Circle No. 10**

ac-dc Welder

Single machine provides ac or dc welding current at flip of switch on P&H DA-200. The *Harnischfeger Corp.* welder operates on 220/440 v single phase current, with service ac welding range of 7 to 275 a or 7 to 200 a dc. **Circle No. 11**

News and Notes

Clark Equipment Co. announced acquisition of inventories, engineering designs, trademarks, and certain other assets of the Torcon Corp., producer of torque converters . . . Torcon's capital stock was not involved . . . *Yuba Mfg. Co.*, builder of bucket ladder dredges for nearly 50 years, has re-entered the hydraulic dredge field . . . *Hardinge Co.* has moved its Canadian office to a new address at 44 Victoria St., Toronto 1, Ont.

Free Literature

(21) STEEL BUILDINGS: *Armco Drainage & Metal Products Inc.* 40-ft interlocking steel paneled buildings are now available in heights up to 20 feet. Bulletin shows how units can be placed side by side, making the total floor area to be enclosed virtually unlimited both as to width or length.

(22) REFRACTORY CONCRETE: Widely used to withstand service temperatures up to 2600°F or more, refractory concrete is the subject of a booklet from the Lumnite Div., *Universal Atlas Cement Co.* Given are special properties, mix proportions and aggregates, industrial applications, and other data.

(23) VIBRATING SCREEN: A 16-page bulletin No. 651 from *Pioneer Engineering Works Inc.* covers the improved Mesabi vibrating screen, built specifically for severe service applications. Installation drawings, data, and specifications, text material, and tables enable proper selection of the correct screen for the job.

(24) DRILLS, SHOVELS, CRANES: *Bucyrus-Erie's* general catalog GC-5A describes a complete line of excavating, drilling, and materials handling equipment. The 70-page, 2-color catalog features concise copy, condensed specifications, and more than 100 photo-illustrations of machines on specific jobs.

(25) CONVEYOR BELT CLEANER: *Stephens-Adamson's* bulletin 651 gives complete details on the S-A spring type conveyor belt cleaner, designed to remove material adhering to the belt after discharge, thus insuring maximum belt life by preventing cover wear. Steel wiper blades perpendicular to the belt surface, but diagonal to its line of travel scrape off material, allowing it to fall to discharge chute.

(26) CLASSIFIED INDEX: *Minneapolis-Honeywell Regulator Co.'s* bulletin 100-C lists all current Honeywell Industrial Div. literature. Numbers and titles of all catalogs, bulletins, specification sheets, and instrumentation data sheets are included.

(27) MINING EQUIPMENT: Crawler tractors, tractor shovels, motor graders, motor scrapers, motor wagons, pull scrapers, and power units all have a place in the mining industry to increase output and

profits. The versatility of these *Allis-Chalmers'* units is illustrated in "Modern Mining Methods," a 2-color, 16-page booklet.

(28) HYDRAULICKING: Folder from *Intelli-Giant Div., Chiksan Co.* describes how Intelli-Giant by utilizing the force of the water itself eliminates stabilization weight required in former machines and increases mobility and maneuverability. Coverage is 320° horizontal and 120° vertical; water pressure 30 to 300 lb; weight of model MH is 800 lb.

(29) PORTABLE EARTH DRILL: *Soiltest Inc.* has bulletin on the McCulloch high speed portable earth drill for fence building, horizontal boring, planting anodes, erecting signs, setting poles, and for converting into a chain saw. This two-man drill weighs only 87 lb.

(30) TUNGSTEN: *Sylvania Electric Products Inc.* brochure devoted to the manufacture, properties, and uses of tungsten has colorful flow chart showing how Sylvania tungsten is handled from ore to finished product.

(31) FIRE PREVENTION: *Gardner Laboratory's* bulletin 157 deals with Justrite safety cans for handling highly volatile liquids, oily rags, and other flammable material.

(32) ELECTRICAL MAINTENANCE: As more and more U. S. plants use automatic control, emphasis shifts from the production worker to the maintenance worker. "Five Steps to Productive Maintenance" from *General Electric Co.* provides detailed information on organizing to meet the demands of this trend.

(33) ALUMINUM WELDING: *Alcoa* experts spent over a year compiling the 176-page book, "Welding Alcoa Aluminum." This comprehensive, well-illustrated manual is divided into 14 chapters devoted to various methods of welding, pressure welding, weld performance, quality control, safety, and important data.

(34) EARTHMOVING: A two color, 48-page catalog from *International Harvester Co.* presents the complete line of International industrial power products, 73 in all. Described are crawler tractors, rubber tired tractors with scrapers, hydraulic and cable-controlled bulldozers, bull-graders, angle bulldozers, attachments, gasoline and gas power units.

(35) STORAGE BATTERY LOCOMOTIVES: Bulletin M-549 from Mancha Storage Battery Locomotive Div., *Goodman Mfg. Co.* covers "efficient, trouble-free, low cost haulage" for mine, tunnels, and industrial plants.

MAIL THIS CARD

for more information on items described in Manufacturers News and for bulletins and catalogs listed in the Free Literature section.

Mining Engineering **29 West 39th St.** **New York 18, N. Y.**

Not good after February 15, 1955 — if mailed in U. S. or Canada

Please send me { More Information ☐ / Price Data ☐ / Free Literature ☐ } on items circled.

1	2	3	4	5	6	7	8	9	10
11	12	13	14	15	16	17	18	19	20
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51	52	53	54	55	56	57	58	59	60
61	62	63	Students should write direct to manufacturer.						

Name_____ Title_____

Company_____

Street_____

City and Zone_____ State_____

(36) FOR SKEPTICS: *National Safety Council's* "What's in It For Me?" uses few words and plenty of cartoons to put across its message that no one is immune to accidents. It's not where you work, but how you work that makes for safety.

(37) INDUSTRIAL PRODUCTS: *American Brake Shoe Co.* has a catalog with emphasis on industrial products. The 48-page, two-color book illustrates representative parts produced by 11 divisions of the company and details physical properties or characteristics where pertinent. Breakdown is by type of product, with sections on ferrous castings, nonferrous castings, bearing materials, sintered metals, steel forgings, welding products, air compressors, pumps, and railroad products.

(38) CARBIDE BITS: *Manchester Bit Corp.'s* folder on custom carbide insert rock bits covers: jackhammer, stoping, heavy drifter and wagon drill, and large bit series, as well as special bits and Manchester's consultant service.

(39) SHALLOW SEISMIC SURVEYS: The High Resolution reflection seismograph system, described in *Houston Technical Laboratories* bulletin, now makes possible accurate reflection survey in shallow depth ranges of 100 to 2500 ft. This portable equipment can be used to determine thickness of gravels, give valuable data on depth and extent of overburden and buried channels, and locate shallow mineral beds or seams, dikes, and similar features.

(40) COAL CRUSHERS: Special construction elements, speeds, and dimensions are given in the 4-page bulletin from *American Pulverizer Co.* of St. Louis on the WC and WS series of rolling ring coal crushers. Capacities range from 15 tph, 85 to 95 pct passing ¼-in. sq openings, for the WC-12 crusher, up to 188 tph, same openings, for the WS-40 crusher.

(41) PACKINGS & GASKETS: Chempac materials, virtually unaffected by corrosive chemicals and active solvents, are made of asbestos treated with Du Pont's Teflon. *Johns-Manville* in "Chempac Packings & Gaskets" shows how to reduce shutdowns in chemical and process equipment.

(42) TRIANGULATION: The famous Universal Theodolite Wild T2, according to *Wild Heerbrugg Instruments Inc.*, has been completely redesigned. While retaining the same compactness as before, the new Swiss-made telescope is said to be practically free of chromatic aberration, even though the image remains perfectly sharp and plane beyond the stadia marks. Well-illustrated bulletin also contains data on other surveying instruments.

(43) MINING MACHINERY: Bulletin 541 is available from *Morse Bros. Machinery Co.*, now celebrating its 56th year of service. The 64-page catalog lists mining and milling machinery and electrical, industrial and construction equipment.

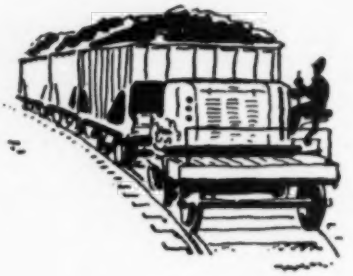
(44) OIL & GAS BURNERS: *Hauck Mfg. Co.'s* catalog 414 covers the series 480P burners for rotary sand and stone dryers and rotary kilns. Using low pressure air and burning oil or a combination of gas and oil, it is reported better results can now be obtained with the long narrow flame of this new series.

(45) TORQUE CONVERTERS: *Caterpillar Tractor Co.* lists eight money making features in "Cat Diesel Engines Plus Hydraulic Torque Converters." Included are descriptions of equipment applications and a complete chart of Cat diesel engine power ratings.

(46) RUBBER CATALOG: *Raybestos-Manhattan Inc.* catalog 25-C features such new products as the R-M Super-Power V-belt, the XDC conveyor belt cover, and the Poly-V drive, which combines the simplicity of flat belts and grip of V-belts.

(47) TESTING SIEVES: *Newark Wire Cloth Co.* has a bulletin on its line of standard testing sieves, market grade testing sieves, and the Newark testing sieve shaker.

(48) JEEP POWER PLANTS: Four 4 and 6 cylinder Power Giant engines are covered in a brochure from the industrial engine dept. of *Willys Motors Inc.* Specifications,



horsepower, torque, fuel consumption figures, and other data are given on the Jeep, Hurricane, Lightning, and Giant Hurricane.

(49) SHAFT MOUNTED DRIVES: Complete information on high ratios of speed reduction in limited space is contained in bulletin 7101 from *The Falk Corp.* It will interest those who use or design machines requiring input speeds between 420 and 10 rpm in the ½ to 30 hp range.

(50) BELT CONVEYOR: For bulletin 132-B on *Hewitt-Robins* belt conveyor. See page 1043.

(51) PREFABRICATED DOCK: For illustrated booklet describing advantages offered by the *De Long* dock. See page 1052.

(52) CRUSHING DATA: For *Allis-Chalmers'* 32-page book. See page 1050.

(53) LOWER GRINDING COSTS: For *International Nickel Co.'s* bulletin, "Engineering Properties and Applications of Ni-Hard." See page 1048.

(54) REDUCTION CRUSHER: For bulletin 7112 from *Traylor Eng. & Mfg. Co.* See page 1060D.

(55) JET SIZER: For *Dorr Co.'s* bulletin 2342 on the Dorrco jet sizer. See page 1060A.

MILLING EQUIPMENT: For *Denver Equipment Co.'s* item 311 described on page 1062 circle (56). Circle (57) for item 313; (58) for item 315; (59) for item 317. For items on the opposite page, circle (60) for item 312; (61) for item 314; (62) for item 316; (63) for item 318.

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Books for Engineers

The Orange Free State Gold Mines, by Paul Klempner, *P. Klempner,* 20 Mowbray Road, South Shields, County Durham, England, about $3.00, 127 pp., 2nd ed., 1954.—Information and carefully prepared data on new goldfields with a view to giving a fair assessment of the potential possibilities of Orange Free State gold mines from an investor's point of view. In this new edition, Mr. Klempner, well-known South African consulting engineer, shows how his calculations have been developed and how his graphs have been constructed. To keep this information up-to-date, supplementary information on each mine will be issued. A subscription to these quarterly supplements is about $3.00.

Chemical Business Handbook, edited by John H. Perry, *McGraw-Hill Book Co. Inc.,* $17.00, various paging, 10⅛x7⅝ in., bound, 1st ed., 1954.—This book, designed primarily for technical men in executive positions as a basis for training programs and reference source, can also serve as a text for courses in business administration. The 20 sections, prepared by specialists, cover major phases of business enterprise—finance, market research, purchasing, sales. The material is treated in such a way as to be applicable to industries other than chemical.

Metal Statistics 1954, *American Metal Market,* $3.00, 848 pp., annual edition, 1954.—This 47th annual edition of a well-known compilation includes information similar to that of previous issues—historical and current statistics on ferrous and nonferrous metals and on related subjects.

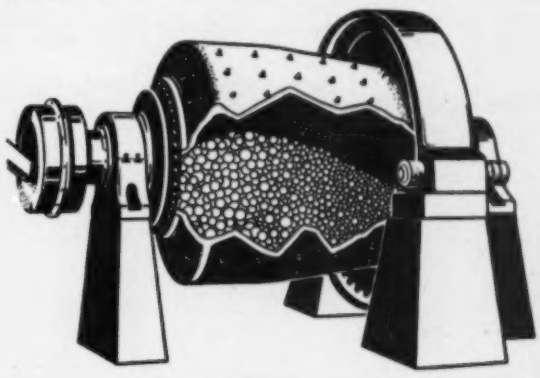
The Technical Report, edited by B. H. Weil, *Reinhold Publishing Corp.,* $12.00, 485 pp., 1954.—Based, in part, on the 1953 Symposium on the research report held by the American Chemical Society, this book discusses both the theory and practice of writing, illustrating editing, duplicating, filing, retrieving, and using technical reports. A chapter is devoted to the oral report, and the question of security-classified reports is considered. References.

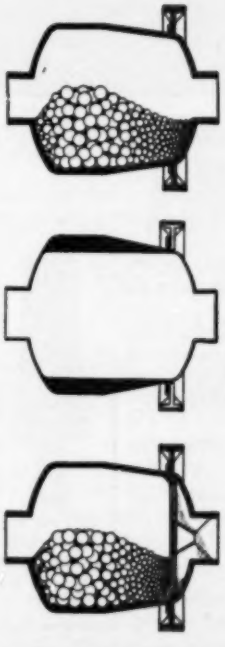
A Textbook of Metallurgy, by A. B. Bailey, *St. Martin's Press,* $8.00, 560 pp., 1954.—Surveys the nature, structure, and properties of metals and alloys, ores and their preparation, extraction and refining techniques, the manufacture of ingots and production of castings, mechanical working processes, mechanical and nondestructive test methods, and pyrometry. The book is intended for students preparing for degree or professional examinations and includes lists of additional readings and representative questions.

ORDER YOUR BOOKS THROUGH AIME—Address Irene K. Sharp, Book Department. Ten per cent discount given whenever possible. Order Government publications direct from the agency concerned.

Diesel Locomotives, by John Draney, *American Technical Society,* Chicago, $4.85, 344 pp., 1954.—Basic information for training personnel building, operating, and servicing diesel-electric locomotives. Fundamentals of engine cycles and of combustion are treated briefly and separate chapters give details of engine parts and accessories. The latter half of the book discusses electrical fundamentals. Material included is a revision of a 1943 text issued in two parts.

ASTM Standards on Cement, *American Society for Testing Materials,* Philadelphia, $2.75, 247 pp., 1954.—A useful compilation of 34 test methods, specifications, and definitions for natural, portland, and masonry cement.





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Correct ball segregation without the use of special liners. Segregation maintained throughout entire lining life—minimum scrap loss. The mill shape does it. Peripheral speed highest at coarse crushing end—lowest at fine grinding end.

Less floor space required than for cylindrical mill of equal capacity. Greater capacity for same floor space. Tricone has 15% to 20% greater volume for given floor space.

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Production of High-Btu Oil Gases from Crude Shale Oils, by E. B. Shultz, Jr., J. J. Guyer, and H. R. Linden, *Institute of Gas Technology,* 17 West 34th St., Technology Center, Chicago 16, $2.50, 23 pp., February 1954.—This interim report presents experimental data obtained in the thermal gasification of two typical crude shale oils in a 2.5-in. ID externally heated laboratory cracking tube, over a range of cracking conditions resulting in the production of gases with heating values from 900 to 1500 Btu/SCF.

Lubricating Oils for Industrial Engines, *Internal Combustion Engine Institute,* 201 North Wells St., Chicago 6, Ill., 20¢, 48 pp., June 1954.—A list of trade names of more than 1200 lubricating oils grouped in accordance with their ability to meet three types of service requirements common to industrial engine operation. Performance levels are based upon information received directly from the oil suppliers.

United Nations Tin Conference 1950 and 1953, Summary of Proceedings, *United Nations,* available from Columbia University Press, 2960 Broadway, New York 27, N. Y., 30¢, 26 pp., May 1954.

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Mining operations are crowded at best—there's nearly always too much equipment to move in or out of the hole.

That's why CF&I Rock Bolts are so popular with mine operators. Because these bolts are embedded in the walls and ceiling, they afford excellent support without taking up valuable space required by old-fashioned timbering methods.

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Field Soniscope Tests of Concrete; Report No. 1, 1953 Tests, Technical Memorandum No. 6-383, 50¢, 21 pp., April 1954. **Permeability and Triaxial Tests of Lean Mass Concrete,** Technical Memorandum No. 6-380, $1.00, 87 pp., March 1954. **Investigation of Durability of Concrete Exposed to Natural Weathering, Report No. 5, Summary of Results 1936-1953,** Technical Memorandum No. 6-226, $1.00, 72 pp., May 1954. Available from Director, *Waterways Experiment Station, Corps of Engineers, U. S. Army,* P. O. Box 631, Vicksburg, Miss.

The Hopper, *Oklahoma Geological Survey,* Norman, Okla. A monthly publication devoted to Oklahoma's development. Mailed free of charge.

The Lower Kittanning No. 5 Coal Bed in Ohio, by Russell A. Brant, Report of Investigations No. 21, *Ohio Dept. of Natural Resources,* Orton Hall, Ohio State University, Columbus 10, 50¢, 59 pp., illus., maps, 1954.—This report reveals that at least 9.9 billion tons of bituminous coal were present in the Lower Kittanning bed in Ohio before mining began. However, under present mining and economic conditions only about 10 pct or 1 billion tons are available as mineable coal. These reserves are located mainly in or adjacent to Stark, Perry, and Lawrence Counties.

What Every Engineer Should Know About Rubber, by W. J. S. Naunton, *British Rubber Development Board,* London, available in the U. S. from Natural Rubber Bureau, 1631 K St., N. W., Washington 6, D. C., 50¢, 128 pp., illus., 1954.—Written entirely with the engineer's working day in mind, this small, but excellent book condenses a broad cross-section of information on rubber. The author, consultant on rubber and plastics to the Admiralty, believes that it is important for the engineer to know not only how rubber components are used but also how they are made. "The facts are set out with no attempt to boost virtues or conceal defects; after all, the material with no shortcomings has still to be discovered!"

Prospecting With A Counter, by Robert J. Wright, *Supt. of Documents, U. S. Government Printing Office,* Washington 25, D. C., 30¢, 68 pp., 1954.—The author is chief of the Geologic Branch, Exploration Div., Atomic Energy Commission, Grand Junction, Colo. His booklet, a revised version of an earlier AEC report, RME-4028, describes the use of radiation detection instruments in prospecting for uranium ores. It is a companion reference to **Prospecting for Uranium,** 55¢, also sold by the Supt. of Documents.

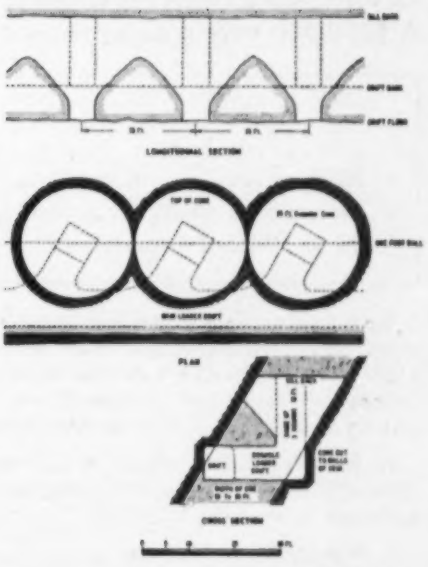
INCREASE PRODUCTION LOWER COSTS

Yes! You can increase production and lower costs by using Eimco loaders to load your production tonnage.

The advantages of being able to load larger pieces, use less powder, practically eliminate secondary blasting and absolutely eliminate expensive chutes and grizzlies, will enable your mine to get into production in a new area faster at less expense.

That's why so many mining men are traveling to see mines that have changed their systems to production loading with Eimcos.

Many different ideas for saving time and lowering costs have been developed by operators to fit their particular conditions. A sketch of one of these is at right. Eimco engineers have helped work out numerous systems, they will be glad to help you. Write Eimco for information.



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Our mills...like yours aim for Lower Grinding Costs

Results obtained during actual operations in our Copper Cliff plant show economies you can anticipate for your own mills under like conditions.

A $300,000 saving during the past ten years is represented by the following data on liner performance in 6½′ x 15′ open end rod mills (2½″ diameter rods).

Typical figures showing the superiority of Ni-Hard® over manganese steel liners are tabulated:

	Operating Hours per set liners	Dry tons of ore ground per set	Lbs. of metal used per ton of ore ground
Ni-Hard	12,921	541,981	.0988
Manganese steel	7,061	250,498	.1739
% of Ni-Hard Superiority	84.2%	116.4%	76.0%

How long do your grates last? At the Copper Cliff concentrator, illustrated, 6½′ x 15′ Marcy mills using 1½″ Ni-Hard grinding balls are equipped with 54″ diameter Ni-Hard half circular grates. These grates give an average life of 8000 to 9000 hours.

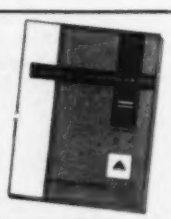
At another INCO concentrator . . . a test on five impellers for 6″ slurry pumps handling rod and ball mill discharge showed the following:

Average life of Ni-Hard . . . . . 846 hours
Average life of white iron . . . . . 449 hours

These are typical . . . not exceptional . . . examples. They illustrate the superior abrasion resistance offered by Ni-Hard, and the way it keeps overhead down.

Ni-Hard can also help you the same way it helps us in other applications such as scoop lips, roll crusher feeder and cheek plates, flotation hoods and particularly, grinding balls. If you have abrasion problems, let INCO's experience help you solve them.

The completely revised third edition of "Engineering Properties and Applications of Ni-Hard" is now available. Write for your copy today.



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THE ATOMIC ENERGY COMMISSION announced a $67 million program to expand uranium processing facilities in three states. A $33.3 million plant 27 miles west of St. Louis will be the largest single project under the program. The plant is to be operated under government contract by the Mallinckrodt Chemical Works. An additional $6.5 million will go into expansion of existing facilities already operated by Mallinckrodt. The commission also plans to spend $20.1 million at Fernald, Ohio, for modifications and existing buildings and equipment.

THE FOREIGN OPERATIONS ADMINISTRATION has been buying coal in areas classified by the Government as distressed for shipment overseas under aid programs. Foreign Aid Director Harold Stassen stated that his organization has set a goal of 10 million tons of coal to be sent abroad. Until now, the sole purchases were by foreign countries that had been given credits by the U. S.

Aluminium Ltd. announced that capacity of its Kitimat, B. C., smelter will be enlarged by 60,000 tons of ingots per year at a cost of $45 million. The 65 pct increase was decided upon after market studies indicated its justification.

NICKEL PRODUCTION in the U. S. may increase from the 700 tons produced in 1953 to 2500 tons in 1954. In the face of nickel consumption last year of 106,000 tons the U. S. is producing only 1 pct of its total requirements. However, the Senate Interior Committee on the Accessibility of Strategic Materials to the U. S. in Time of War and For Our Expanding Economy thinks that the U. S. may produce as much as 10 pct of its nickel needs by 1955. Nickel production in the Western Hemisphere was 157,000 tons in 1953. Current production is estimated to be at the rate of 170,000 tons.

Freeport Sulphur Co. is reported to be negotiating with Pittsburgh Consolidation Coal Co. for a joint $16 million potash project near Carlsbad, N. M.

THE FIRST COAL to be mined since May in the Panther Valley mines at Lansford, Pa., was produced recently as miners went back to pits leased from Lehigh Coal & Navigation Co. by Panther Valley Coal Co. Some 1000 men went back to work after signing an agreement to work harder for greater coal production. The agreement supplemented the regular United Mine Workers Union contract. Panther Valley Coal expects to produce 5500 to 6000 tons of coal per day, employing about 1300 of the valley's 5000 miners.

Kennecott Copper Corp.'s Braden mine at El Teniente, Chile, has been placed on a seven-day basis in an effort to cope with the copper shortage. Chino is also working seven days and Kennecott is reported to be considering lengthening the work week at its other operations. However, at the time the seven-day week was instituted, the Utah mine was working only four days due to a strike at the Garfield refinery.

A NEW ENTERPRISE, Stanleigh Mining Corp. Ltd. of Toronto, has acquired a block of claims in the past year that parallels major showing of Algom in the Blind River region. Howard S. Strouth of Standard Ore & Alloys Corp., New York, holders of a major interest in the new company, heads the Canadian firm. Stanleigh, capitalized at $5 million, plans to spend $250,000 for the present drilling program. Another U. S. firm, Stancan Uranium Corp. (no connection with Standard Ore & Alloys) is entering the Canadian uranium search. Mr. Fraser is one of the directors.

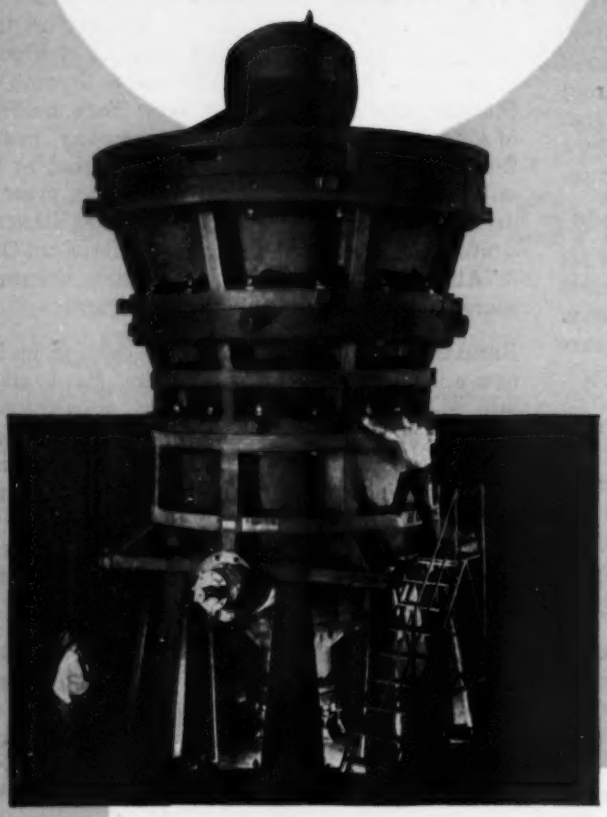
BETHLEHEM STEEL CO. expects production at its new mines 14 miles south of Reading, Pa., to start late in 1957. Work on the two Morgantown shafts, one 1600 ft and the other 1200 ft, has been impeded by water and other adverse conditions. About 12,000 tons of iron ore is expected to come from the mines daily when full production is reached, sometime in 1960. Bethlehem is reported to be putting about $34 million into the operation.

CLEVELAND-CLIFFS IRON ORE CO. and the Island Creek Coal Co., Huntington, W. Va., will be associated in the coal business through the sales of Cliffs' coal marketing division and the Duluth and Port Huron Docks to Island Creek. Also involved was the sale through Island Creek of the Green Bay, Wis., dock and Cliffs' interest in the Escanaba Coal & Dock Co. to the C. Reiss Coal Co. of Sheboygan, Wis.

Office of Defense Mobilization announced that Government will spend another $2.5 million for strategic and critical materials to meet minimum stockpile objectives. Arthur S. Flemming, ODM director, announced that his agency plans to move from minimum to long-range objectives whenever "we can acquire materials at favorable prices and whenever we can help to strengthen our domestic mobilization base."

SUPERIOR Gyratory CRUSHERS

29 in 3 YEARS!



SUPERIOR primary and secondary gyratory crushers have been making a big name for themselves since they were introduced three years ago. Twenty-six are installed and operating... three are under construction now, including a huge 60 x 109 machine — largest crusher the world has ever known.

A policy of simplifying design and controlling quality has made Allis-Chalmers the leading builder of crushers. A continuing policy of *improving* crusher design has greatly extended this leadership.

This vast backlog of crusher application experience — over 75 years of it — is always available to you when you want to make sure of a successful installation. Allis-Chalmers, Milwaukee 1, Wisconsin.

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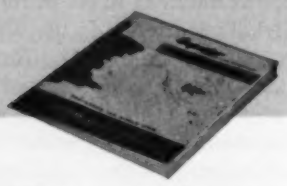
Superior is an Allis-Chalmers trademark.



NEW 32-Page Book Contains Helpful Crushing Data

- Packed with factual "how to" information on figuring hp requirements, impact and compressive strengths.
- Step-by-step procedures for estimating gyratory crusher sizes, capacities. Examples are worked out.
- Many other valuable facts on gyratory crusher operation . . . application . . . engineering.

It's a book you'll want to have and keep!

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Mining Men, Public View AMC Show in San Francisco

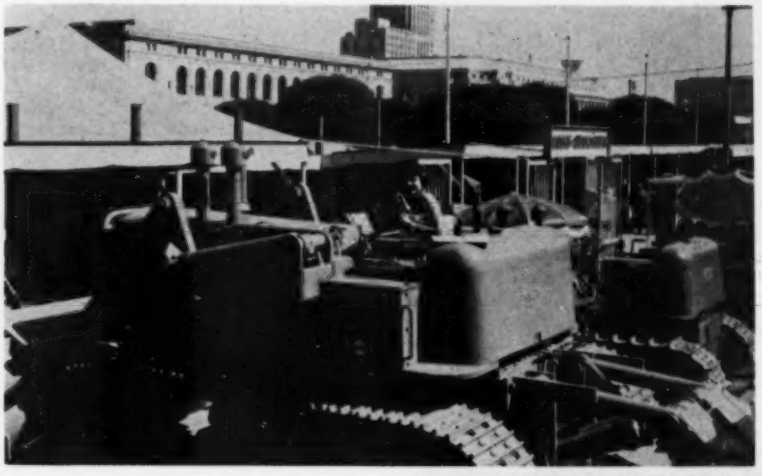
The American Mining Congress Convention and Exposition at San Francisco attracted some 8000 persons including 1300 ladies during its five days in San Francisco. Exhibits took up some 33,500 ft of floor space in the Civic Auditorium and overflowed outdoors where another 32,500 sq ft were covered with metal and nonmetal mining equipment.

The last metal and nonmetal mining show was held in Denver two years ago, but the 1954 version was bigger by about 15 pct. Of those who attended the show, about 3500 were directly connected with the industry. The remainder of the show's visitors came from the ranks of the general public who wanted to see what mining was all about.

Technical sessions covered everything from titanium, to uranium, to drilling, to economics, to safety, to legislation. Leonard R. Flicker, senior safety engineer, Permanente

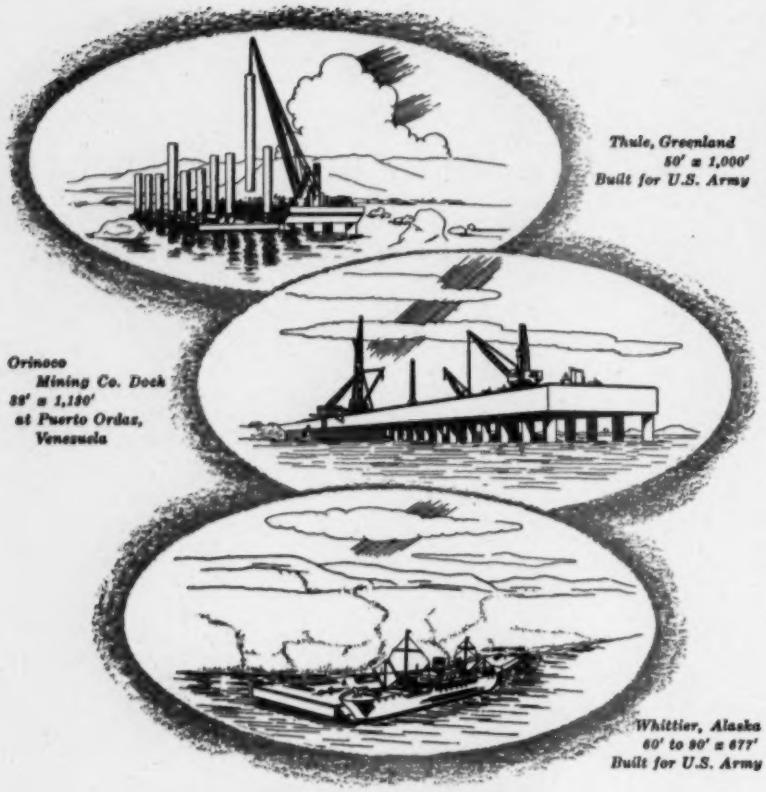
(Continued on page 1052)

Equipment displays overflowed Civic Auditorium floor space and occupied an additional 32,500 sq ft. The Civic Auditorium can be seen in the background above.



Allis-Chalmers, at left, was on hand to show the latest in its diesel powered tractors, complete with pretty receptionist.

Civic Auditorium floor space, some 33,500 sq ft, was covered with all manner of nonmetal and metal mining equipment. Not only miners were attracted to the displays. The lay public flocked to get a glimpse of the industry.



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is the prefabricated dock that solves tough problems

De Long Docks can be erected speedily under adverse conditions such as swift, turbulent waters, or heavy tidal action. No special foundation preparation is necessary. The 6' diameter steel caissons may be keyed into rock where little or no overburden exists or driven into any type or any depth of subsoil.

Economically fabricated in shipyards or steel fabricating plants to required size for specified loads, towed to site and raised to desired elevation on its steel caissons, De Long Docks offer many advantages. Dock sections while under tow carry their own caissons and the De Long Airjacks that make the erection of this type of structure possible, as well as all construction equipment and gear required for the field erection operations. Ships can be berthed within a few days after the first De Long section arrives at site. Large docks or piers such as the 1,130 ft. long Orinoco structure are completed within six months after signing of contract including design, fabrication, delivery to site and erection.

De Long Docks can be erected with substantial savings in time and money anywhere in the world, and under any conditions.

Write today for illustrated booklet which describes the advantages offered only by the De Long Dock.

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Mining Show

The Joy Mfg. Co. display had a constant hum of activity about it, as interested delegates asked questions, got answers, and just took a look at the display.

Cement Co., told delegates to the convention that "The true picture of the cost of accidents is seldom realized. Actual costs of an accident can be arrived at by multiplying the compensation and medical costs by five, to account for the many hidden costs not covered by insurance."

Representative Hale Boggs (Dem.-La.) told listeners that they could expect Federal income tax personal exemption to increase by at least $100 as soon as the economy permits. He also said, "I don't believe we will ever see a Federal sales tax . . ." Representative Boggs added that the capital gains tax is regressive and a reduction will be forthcoming.

"The most effective means of encouraging young men to enter the mining industry is to hold out to them the prospect of substantial rewards and of work that will tax their ability to the limit." That's the formula offered by Dean M. P. O'Brien of the College of Engineering, University of California, Berkeley. Marc Boyer, Deputy Minister, Canadian Dept. of Mines & Technical Surveys, Ottawa, termed the growth of the Canadian mining industry since World War II "little short of remarkable." Annual value of mineral production tripled during this period—from $502.8 million in 1946 to more than $1.3 billion in 1953. In spite of this growth, development is in the early stages, Mr. Boyer said.

Block caving is successful as a mining method to be applied to previously stoped ore zones, according to Anaconda Copper Mining Co.'s experience in its Kelley mine in Montana. M. K. Hannifan, assistant general superintendent of the mine, stated that Kelley mine block caving will be extended to an estimated 130 million tons of ore containing 20 lb of recoverable copper per ton and minor amounts of gold and silver.

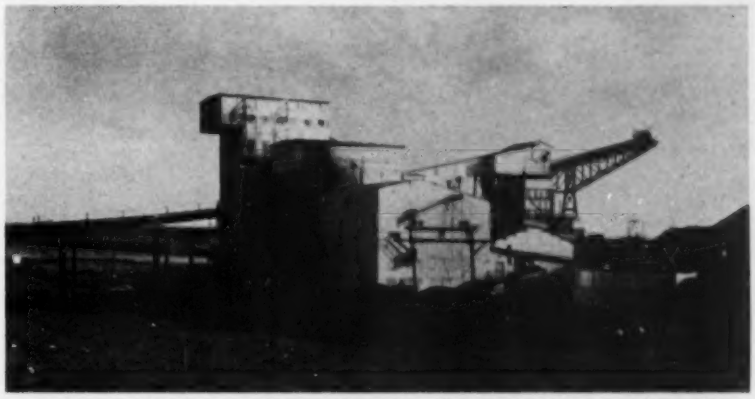
L. E. Snow, general foreman of drilling and blasting for Kennecott Copper Corp., reported that development and use of a special mobile mine drill has been responsible for revolutionary changes at the Utah Copper Div. operations of the company. Since the first mobile drill was used in 1951, eight have been added.

Synchronous Motors Raise Power Factor of Mesabi Plant

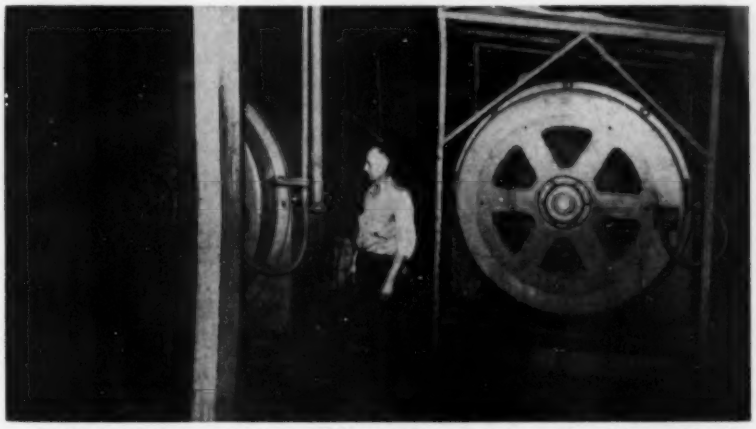
Four iron ore concentrator plants on the Mesabi have been lifted out of the power factor penalty range with monthly savings of $500 by introduction of only two 600 hp synchronous motors.

Since introduction into the electrical system serving the four plants, the two motors have lifted overall power factor from a low of 80 pct to an 86 pct level, safely over the 85 pct penalty limit specified in the company's power contract. Before the new installation all equipment in the four plants was powered by induction motors ranging in size up to 250 hp for the pumps.

Engineers decided to replace two induction motors in one of the plants with a pair of 600-hp Fairbanks-Morse synchronous motors with leading power factor. Although operating alternately, the units were enough to bring power factor of the entire four-plant system up to 86 pct. The motors chosen are designed to operate at constant speed under all load conditions. Each is rated 600 hp, 600 rpm, at 80 pct leading power factor. The power supply is 3-phase, 60 cycle, 2300 v.



Ore from the Mesabi range is processed at this wash plant where the power factor has been raised to 86 pct by two 600-hp synchronous motors, saving the company an average of $500 per month in penalty charges.



Chief electrician inspects the two 600-hp synchronous motors. They drive a pair of 12-in. centrifugal pumps which handle 446,400 gal per hr of tailing. The 600-hp motors are 3-phase, 60 cycle, 2300 v ac.

Each synchronous motor is direct connected to a 12-in. Hydroseal centrifugal pump designed to handle material containing 25 pct solids. These pumps take turns pumping tailing through 7700 ft of 18-in. pipe leading from the plant to the tailing basin three quarters of a mile away. Although 5000 ft of flow is by gravity, pumps have a static lift of 76 ft with a discharge pressure of 90 to 100 psi. Normally the pumps handle 446,400 gal per hr, with solids amounting to 450 LTph.

Government Survey To Cover Minerals

Some 3 million firms in the distribution, manufacturing, and mining industries will be surveyed under the Censuses of Business, Manufactures, and Mineral Industries next year. Congress has appropriated $8.43 million for the job which is to be done by mail. The census will cover 1954 operations. The survey of minerals industries will cover about 35,000 establishments engaged in mining, petroleum production, and quarrying. The information sought will deal with output, payrolls, fuel and power consumption, and other items. Data will be published for states and all counties with important mineral industries. The first such census was taken in 1839 and the most recent in 1939.

In announcing the census, Director Robert W. Burgess of the Bureau of the Census, U. S. Dept. of Commerce, said:

"The Congress has made provision for these important and timely censuses so that up-to-date statistical information may be compiled and published about American industries and trades. The demand for this information has become imperative because the latest available census statistics are from 7 to 15 years old and both Government and private enterprise need fresher information about the nation's many million businesses which serve the needs of a population of nearly 163 million."

Disperse Record Sum From Public Lands

The Bureau of Land Management will distribute a record $11,184,235.43 in revenues from mineral activities from lands it administers to 22 states and Alaska.

Payments cover the first six months of 1954. They compare with payments made from July 1 to December 31 last year of $9,491,155.57. The states received $20,675,791 for the last fiscal year. Revenues from oil and gas leases represent a major portion of the money collected on about 88,000 oil and gas leases on 66 million acres of public domain.

Mineral leares and permits accounted for $59,861,932.39 of the $77,487,465.92 the Bureau received from public land resource activities during the fiscal year 1954.

Biggest slice of the current payment goes to Wyoming, which receives $3,957,508.16. New Mexico is next in line with $2,050,919.10, while California is third with $1,889,846.79. Colorado was in fourth place, receiving $1,528,373.54. Florida, getting only $15.18, ranked last.

CUT LOADING COSTS WITH FAST-RUGGED EIMCO'S

Yes! You cut costs when you use Eimcos for loading.

NOTE these advantages:—

Advantage: Eimcos dig and load materials that are difficult or impossible for other loading equipment.

Reason: Eimcos are designed for tough jobs — digging and loading rough, broken rock. Tracks are designed to oscillate freely even with the loader attachment. The bucket design permits digging in frozen stock piles, rough bottoms, heavy ores and in sticky clay or unbroken conglomerate.

Advantage: Eimcos are more maneuverable.

Reason: Eimcos use independent track control. Separate levers control each track and one track can be run forward while the other runs reverse.

Advantage: Eimcos last longer.

Reason: Torque converter drive is standard on Eimcos. All castings are alloy steel, all construction is extra heavy-duty.

Advantage: Eimcos load faster.

Reason: The overhead principle developed by Eimco is faster. Complete cycle is 10-12 seconds. Shifting from high to low on tractor or loader is done in motion. Shifting from forward to reverse can be done at full speed.

Other Advantages Include: Better visibility with the operator up front. Easier maintenance with clutches that never need adjustment and elimination of all clutches, brakes and gadgets in the final drive.

Let an Eimco engineer show you how you can cut loading costs on the next job.

Eimco 105 with bulldozer attachment

Eimco 105

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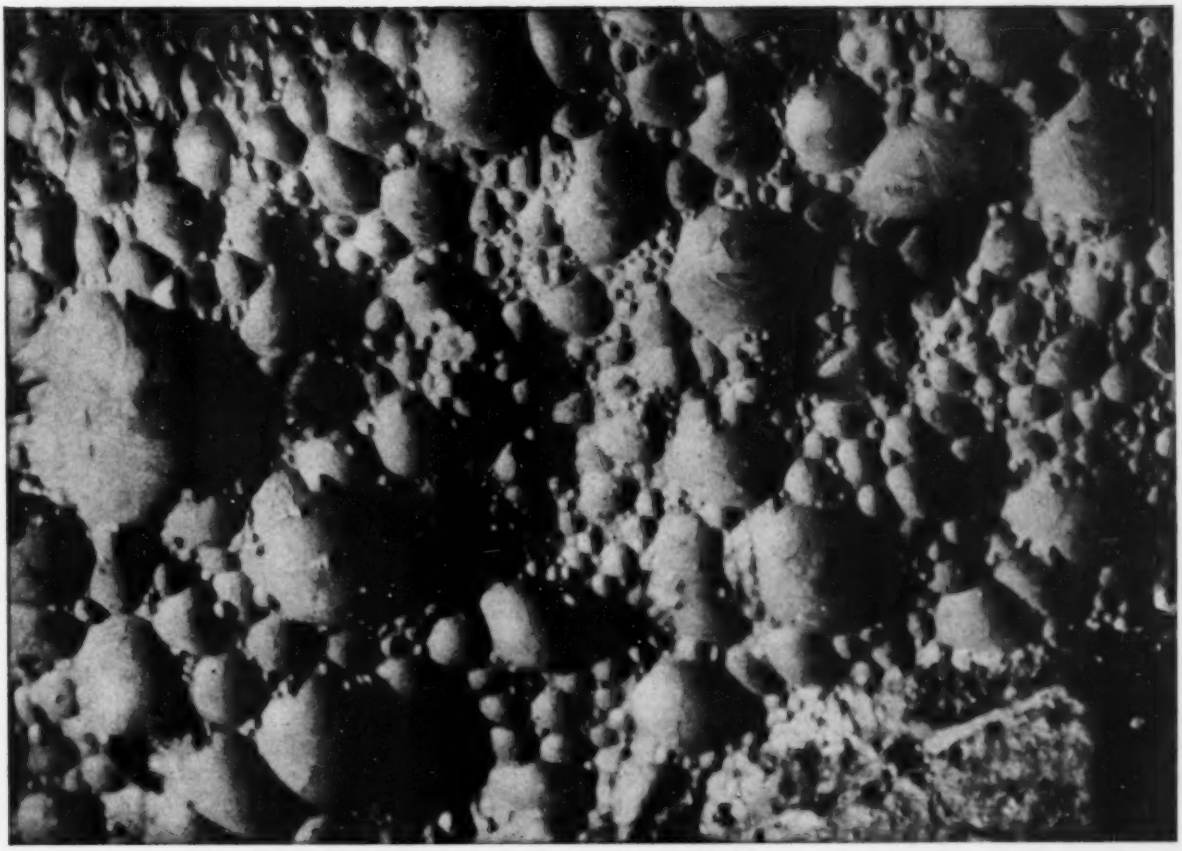

FOR MAXIMUM RECOVERY AT MINIMUM COST

List the characteristics of a good flotation agent and you'll see why Hercules Yarmor® F Pine Oil has remained a standard of quality for more than thirty years. Economy, strong froth with good volume, satisfactory texture, excellent cell-life stability—Yarmor F Pine Oil has them all.

Widely used as a frother for sulfide minerals such as zinc, copper, nickel, iron and lead sulfides, Yarmor F Pine Oil has also shown excellent results in the recovery of gold ores and native metals. This low-cost frother is also ideal for use in the benefication of nonsulfide ores such as coal, mica, quartz, graphite, feldspar, phosphate rock, and talc.

Other Hercules' flotation agents include Hercules Rosin Amine D Acetate, a siliceous mineral collector that is particularly suited for the phosphates. For information on these and other Hercules flotation agents, write:

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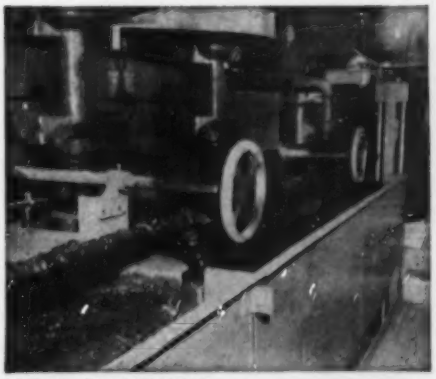


COAL FLOTATION—A growing market for Yarmor is in the recovery of fine anthracite and bituminous from silt.

HERCULES FLOTATION AGENTS

NM54-1

About half the copper produced by the Union Miniere de Haut-Katanga comes from the Leopold mine. The rich sulphide ores of the mine pass through these surface installations. Union Miniere is an outgrowth of Belgian Congo pioneering.

Katanga—Outgrowth of a King's Foresight

Union Minière du Haut-Katanga is one of the largest producers of copper in the world. Deep within the Katanga region of the Belgian Congo, the company's mine and smelter produced some 214,116 metric tons of copper in 1953. Total copper production, since the start of operations at the turn of the century approaches 4.3 million metric tons.

Behind the formation of the Union Minière is the story of a king's foresight and the realization of an American journalist that this was a region of untold possibilities. The King was Leopold II of Belgium, and the American was Henry Morton Stanley who went to Central Africa for *The New York Herald* seeking a man named Livingstone.

Stanley once said, "No students of geography ever paid more attention to my letters . . . to my book, to my lectures on Africa than did King Leopold II." The king dreamed of a mignty civilizing movement and created the Assn. Internationale Africaine. The association organized a number of expeditions which led to explorations that made settlement possible.

Around 1889 the Compagnie du Congo pour le Commerce et l'Industrie organized an expedition under the command of Alexandre Delcommune to explore the Katanga from a scientific and commercial point of view. After being approached by what had by then become the Free State, the CCCI formed the Compagnie du Katanga. In 1900 the Free State and the Compagnie du Katanga formed the Comité Spécial du Katanga, in which both parties vested their rights in the province. While building roads and studying agricultural problems in the area, the Comité made efforts to hasten development of Katanga's mineral wealth.

The Comité on Dec. 8, 1900 granted prospecting rights in South Katanga to Tanganyika Concessions Ltd. This eventually brought about the formation of the Union Minière du Haut-Katanga Ltd. In consideration for an important share of the profit granted to the Comité Spécial du Katanga, Union Minière was granted the right by the latter to mine all the copper deposits within a 7700-sq mile concession, along with all tin deposits within an adjoining concession covering about 5400 sq miles.

Deposits mined at present by Union Minière are at Kipushi, Kolwezi, Musonoi, Ruwe, Lukuni, Luiswishi, Kiswishi, Etoile, Ruashi, and Shinkolobwe.

Copper deposits are in hilly country formed by sedimentary rocks such as shales, limestones, and conglomerates. Most common copper minerals found are malachite, chrysocolla, and the oxide, cuprite. There are other oxides. Sulphide ores are found in depth in certain deposits unaltered by surface waters. The sulphides of the Prince Leopold mine at Kipushi, one of the southeastern mines, contain zinc sulphides intimately associated with copper ores. Separation is brought about by a differential flotation process.

Cadmium is also found in association with the copper-zinc ores of the Prince Leopold mine. The cadmium in the copper concentrates appears in the flue dusts of the water jacketed furnaces at Lubumbashi. Recovery is being studied.

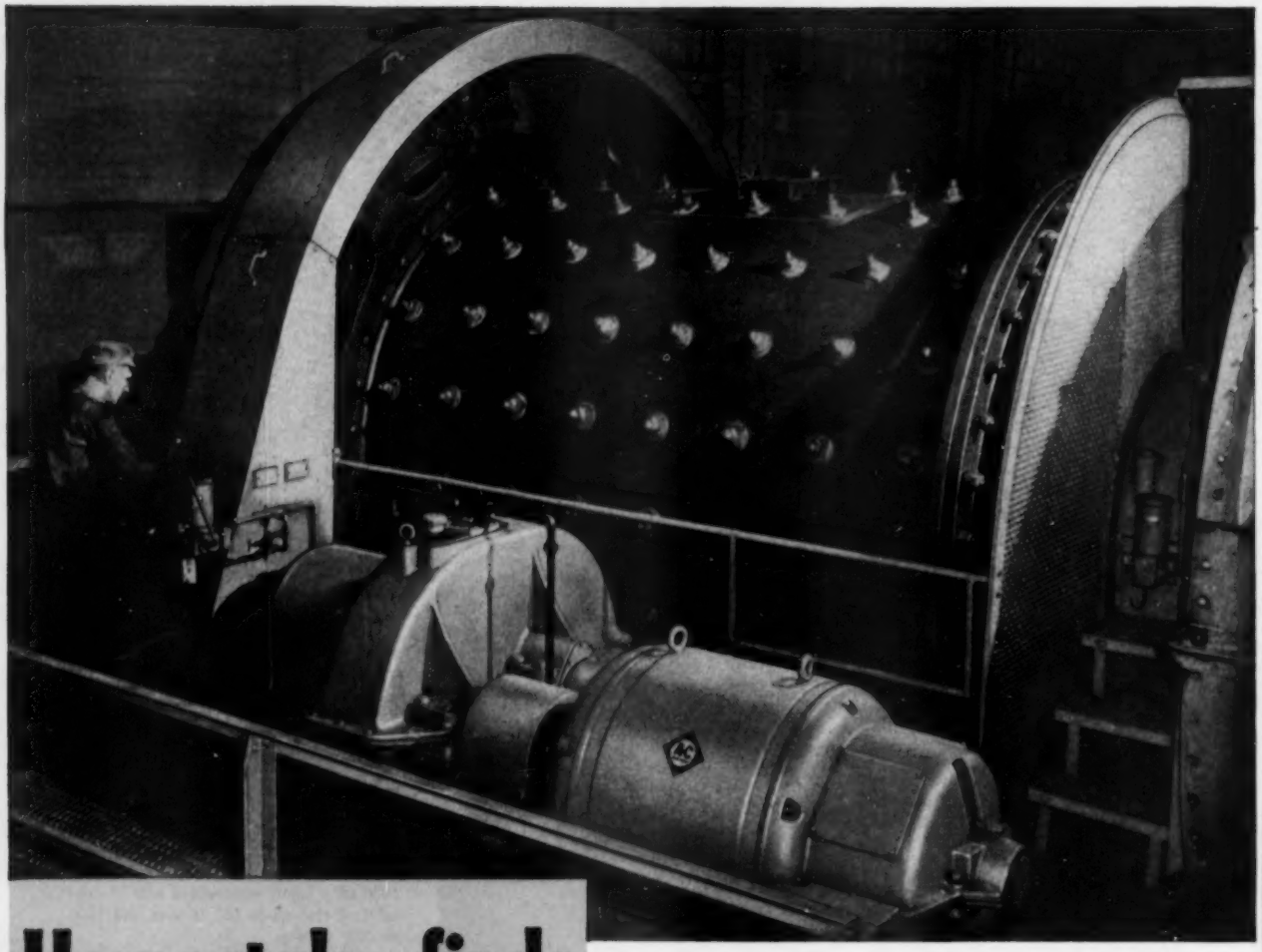
Although it has a tin concession in the eastern part of the Lualaba district, the company is no longer producing tin.

At the Shinkolobwe mine, uranium-radium ore is being mined in the form of pitchblende and its derivatives. The uranium-radium content varies considerably, but is relatively high in comparison to other ores. A plant has been built at Olen, Belgium, by the Société Générale Métallurgique de Hoboken for the treatment of Katanga uranium ores.

The Ruwe mine is about 8 miles northeast of Kolwezi. Gold was discovered there in 1903. It was not until 1937 that the existence of a large deposit of oxidized copper was discovered. To get at the copper a vast stripping operation was undertaken to remove overburden 130 to 165 ft deep. It took nine years to do the job.

The orebody at the Leopold mine is worked by top slicing. Ore is dropped down by gravity to haulage levels and then goes to the hoisting shaft. Stopes are between the 950 and 1150-ft levels. All jackhammers in the mine are equipped with water sprays. The mine has been in operation since 1926 on a highly mineralized vein of copper and zinc 65 to 200 ft wide and 1300 to 2000 ft long.

Part of the ore of the Ruwe mine appears as soft fragments and is easily concentrated by washing. All equipment used in Union Miniere mines is operated by Congolese workers, including large electric shovels, 30-ton capacity dump cars, and electric locomotives.



Homestake finds Gold in MODERNIZATION

Industry's Top Technical Team Helps Reduce Operating Costs

A 2.25 kwhr saving per ton in power consumption — a 37½% reduction in operating labor—a similar saving in maintenance and supply work — considerable space conservation. These benefits realized by Homestake Mining are gilt-edged proof that modernization pays.

Pressured by increasing labor and material costs, in the face of a fixed price on gold, Homestake took a long, hard look at its gold ore operation. With the *help of Allis-Chalmers*, the Homestake staff made a comprehensive study of all phases of ore treatment. Crushing and grinding flow sheets were drawn up and costs estimated. Machines were operated under varying conditions. Power measurements, product analyses and man-hour appraisals were made in a pilot plant run.

As a result of this thorough investigation, a modernization program was launched in each of Homestake's two crushing plants and mill. New equipment was installed and procedures modified to make existing equipment as efficient as possible. Two years of profitable operation have proved that Homestake Mining has struck a rich vein in modernization. A-4470

ALLIS-

ALLIS-CHALMERS
Equipment for the
Mining and Rock
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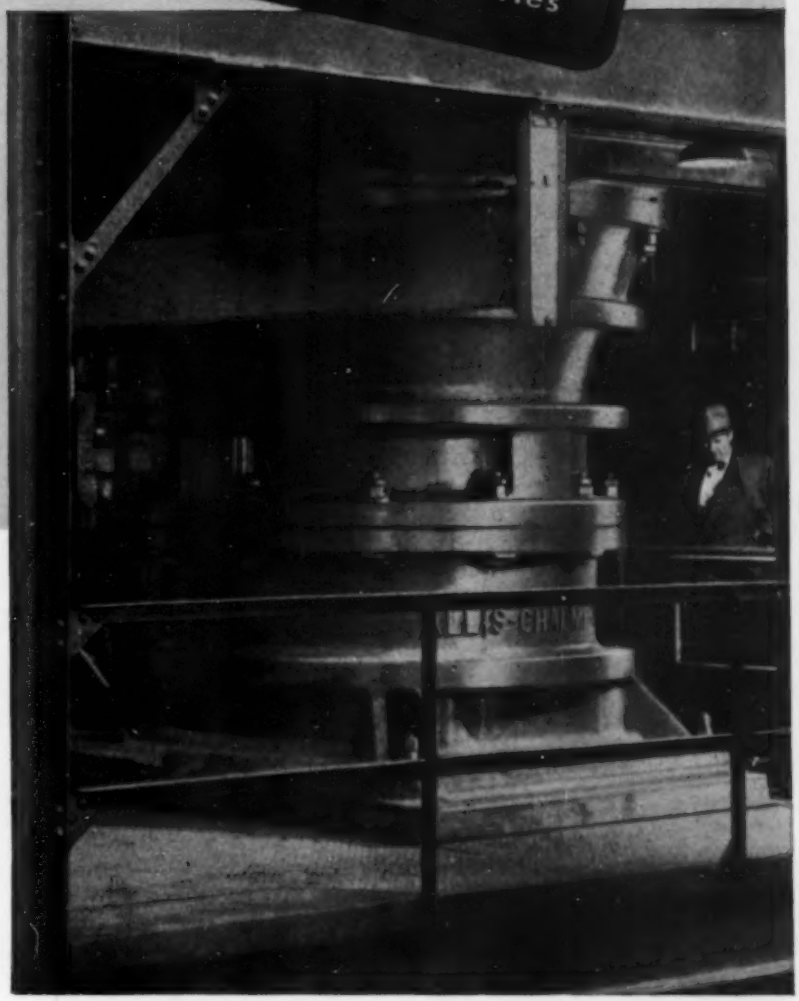

Prominent in the modernization picture at Homestake are three 9 by 11-foot Allis-Chalmers ball mills. These mills, used in the amalgamation process, are equipped with combination drum and double-scoop feeders. Liners are of the Lorain type with hardened lifter bars. Ball additions are 1½ to 2 inches. The mills are driven by 450-hp, 1200-rpm Allis-Chalmers motors through speed reducers.

In each of Homestake's two crushing plants, a modern 30 — 55 *Superior* gyratory crusher has replaced two outmoded 17-inch primary crushers. The new crusher outproduces the combined tonnage of the old crushers and takes up less than half the space. The wearing quality of manganese steel mantles and the strength of short, rigid mainshaft have cut maintenance and parts replacement costs.

Superior is an Allis-Chalmers trademark.

For as Much Help as You Want or Need in Your Modernization Planning, Contact Allis-Chalmers

Your A-C representative, backed by industry's top technical team, is always ready to help you or your consultants. A-C interest in your problems does not end with the installation of equipment. Laboratory facilities, periodic checkups, maintenance and fast emergency parts service are yours continuously.

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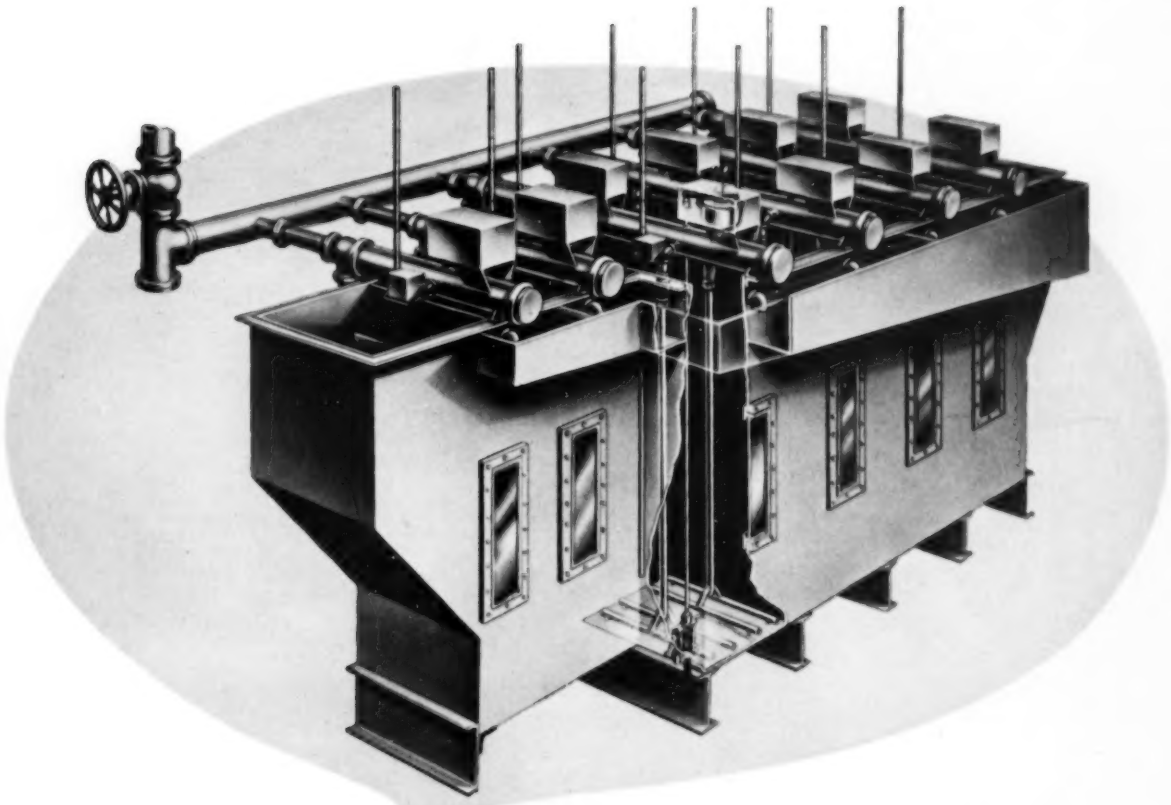
Reno Experiment Station of U. S. Bureau Of Mines Dedicated

At Reno, Nev., on September 25, some 200 guests gathered to dedicate the impressive new Rare and Precious Metals Experiment Station of the U. S. Bureau of Mines. Seated in the bright Nevada sunshine on the lawn in front of the building (referred to by one of the speakers as perhaps the largest patch of grass in the state!), the guests enjoyed a short program which included talks by Minard W. Stout, president of the University of Nevada; J. J. Forbes, director of the Bureau of Mines; Pat McCarran, U. S. Senator from Nevada, (the sudden death of Senator McCarran, a few days later, was a grievous shock to all who knew him); and Felix E. Wormser, assistant secretary of the Interior. Other distinguished guests introduced included: Clifton Young and Walter S. Baring, representatives in Congress from Nevada; Vernon E. Scheid, director of the Mackay School of Mines and the Nevada State Mining Bureau; Jesse Johnson, director of raw materials for the Atomic Energy Commission; T. B. Counselman, vice president of the AIME; Charles W. Mapes, Jr.; Frederick J. DeLongchamps, designer of the building; Steve O'Connor, builder of the structure; Harold C. Miller, regional director of the Bureau of Mines; and Thomas H. Miller, assistant director of the U. S. Geological Survey.

An inspection of the building followed, and then a luncheon at the Riverside Hotel, sponsored by the Nevada Mining Assn. and the Reno Chamber of Commerce. Shortly afterward, Mrs. Felix E. Wormser, president of the Woman's Auxiliary to the AIME, spoke informally before some of the ladies.

John C. Kinnear, Jr., speaks before the large group that gathered for the opening and dedication of the Reno Experiment Station of the U. S. Bureau of Mines. Seated are Thomas H. Miller, Vernon E. Scheid, Steve O'Connor, Walter Baring, Jesse Johnson, H. C. Miller, F. J. DeLongchamps, Charles Mapes, Clifton Young, B. T. Counselman, Felix E. Wormser, the late Senator Pat McCarran, Minard W. Stout, USBM Director J. J. Forbes, Alfred M. Smith, and Major Marvin A. Yardne.

Standing before the marker dedicating the new USBM Experiment Station at Reno are Felix E. Wormser, Asst. Secretary of the Interior; J. J. Forbes, Director of the USBM, and Minard W. Stout, president of the University of Nevada.



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Completely re-designed unit makes more effective use of hydraulic water to produce clean deslimed fractions sized within narrow limits.

For those applications where hydraulic classification is indicated for the sizing or grading of minus 8 mesh pulps the new Dorrco Jet Sizer is the answer.

What's new about the Jet Sizer? First, a new — and more efficient — water distribution system which both reduces initial cost and simplifies maintenance. Second, extreme flexibility of cell arrangement — through which 1½ to 21½ pockets can be arranged in series or in banks in virtually any combination.

Continuous product discharge is controlled by a fully automatic system requiring a minimum of operating attention. Low operating costs result from negligible power requirements and inexpensive low head hydraulic water.

If you'd like more information on the Jet Sizer write for a copy of Bulletin 2342 available at no obligation from The Dorr Company, Stamford, Connecticut.

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FIFTY YEARS 1904-1954

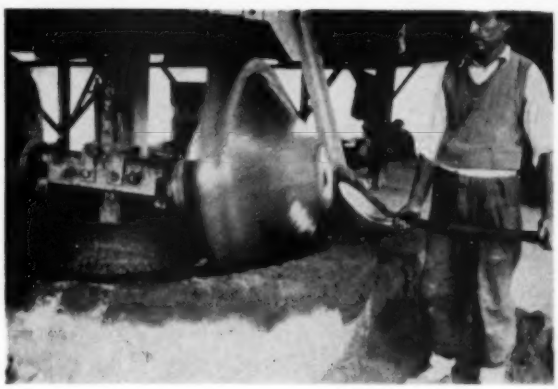
Deep in Chile's Santiago Valley is an excellent example of how rudimentary methods and equipment can achieve low cost-plus-tails . . . with the right reagents.

1

The flow scheme is this simple: Ore running 2% copper carried to the surface (photo 1) is crushed to minus two inches (photo 2) and spread in layers on the mill floor. Over each layer several soup cans of Cyanamid AEROFLOAT® 31 Promoter and AERO* Xanthate 301 are ladled and a dash of pine oil added. Then the batch is spaded and sliced (much as you hand-mix concrete) and shoveled into the Trapiche (photo 3). By adjusting the weir on the trough in which the steel-shod concrete wheels rotate and varying the pulp density, flotation feed overflows at minus 48 mesh to a battery of four roughers and two cleaners (photo 4).

2

At the head of the circuit a booster shot of AERO Xanthate 301 drip feeds through a nail hole in a can, while Sodium Sulfide is added to the last two roughers to float oxide ore. A three-man crew mills 10-12 tons a day. Though control assays are few and far between, smelter returns show a surprisingly high profit.



3

Small or Large... Cyanamid helps

Seven hundred miles north and 8,000 feet higher in the Andes is the mill pictured on the opposite page. It treats upwards of 20,000 tons per day of copper ore assaying substantially the same as the feed to the primitive 10-ton mill above. Though its methods and equipment are ultra modern, for maximum recovery it also uses an efficient reagent combination . . . Cyanamid AEROFLOAT 238 and Isopropyl Xanthate.

4

* Trade Mark

AMERICAN *Cyanamid* COMPANY

mills select the right reagents

At each of these mills a Cyanamid Field Engineer spent many, many hours helping to work out the best reagent set-up for the operating conditions. Whether your mill is large or small, a Cyanamid Field Engineer may be able to help you develop a better reagent combination for your ore on your mill.

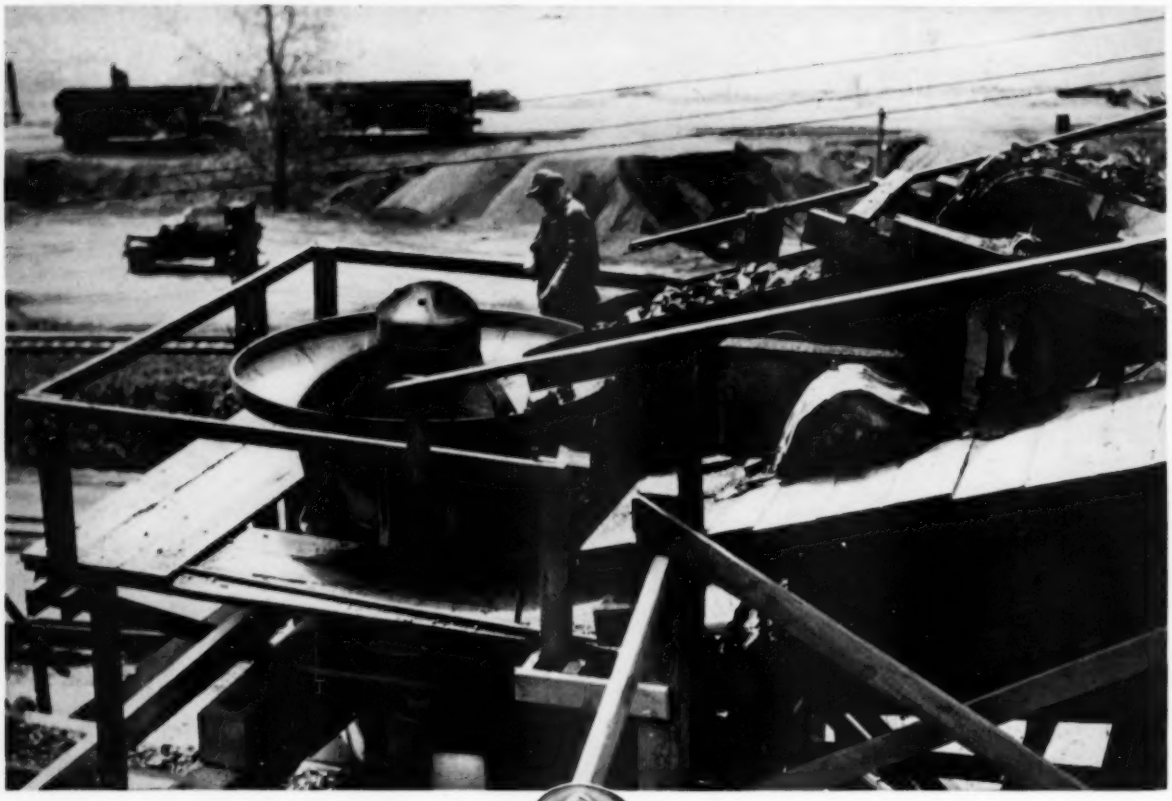
Cyanamid offers a most complete line of flotation and cyanidation chemicals plus efficient processes for separation by specific-gravity differences. A nearby Cyanamid Field Engineer will welcome the opportunity to work with you to cut costs, increase recoveries or both.

MINERAL DRESSING DEPARTMENT 30 ROCKEFELLER PLAZA, NEW YORK 20, N. Y.

more capacity...

that's why more than 806 operators have already bought

TRAYLOR TY REDUCTION CRUSHERS



This cut-away view shows why you get more out of a Traylor TY. Bulletin 7112 gives a complete description of Traylor's original bell head and curved concaves. It will pay you to get a copy. Send coupon today.

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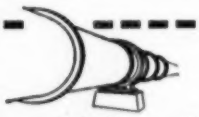
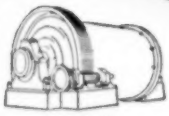
A Traylor LEADS TO GREATER PROFITS

Primary Gyratory Crushers

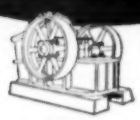


Rotary Kilns

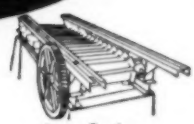
Secondary Gyratory Crushers



Ball Mills



Jaw Crushers



Apron Feeders

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explosives research pays off

The tremendous power of dynamite, skillfully and ingeniously handled, made it economically possible to cross the wide, deep Tennessee River (inset above) with two parallel pipelines, 50 feet apart. Trenches blasted underwater through hard red clay in the shallows, and lime rock in the 40-foot-deep channel, opened the way for speedy dredging operations . . . another example of the efficiency of modern explosives and blasting methods.

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Explosives Department

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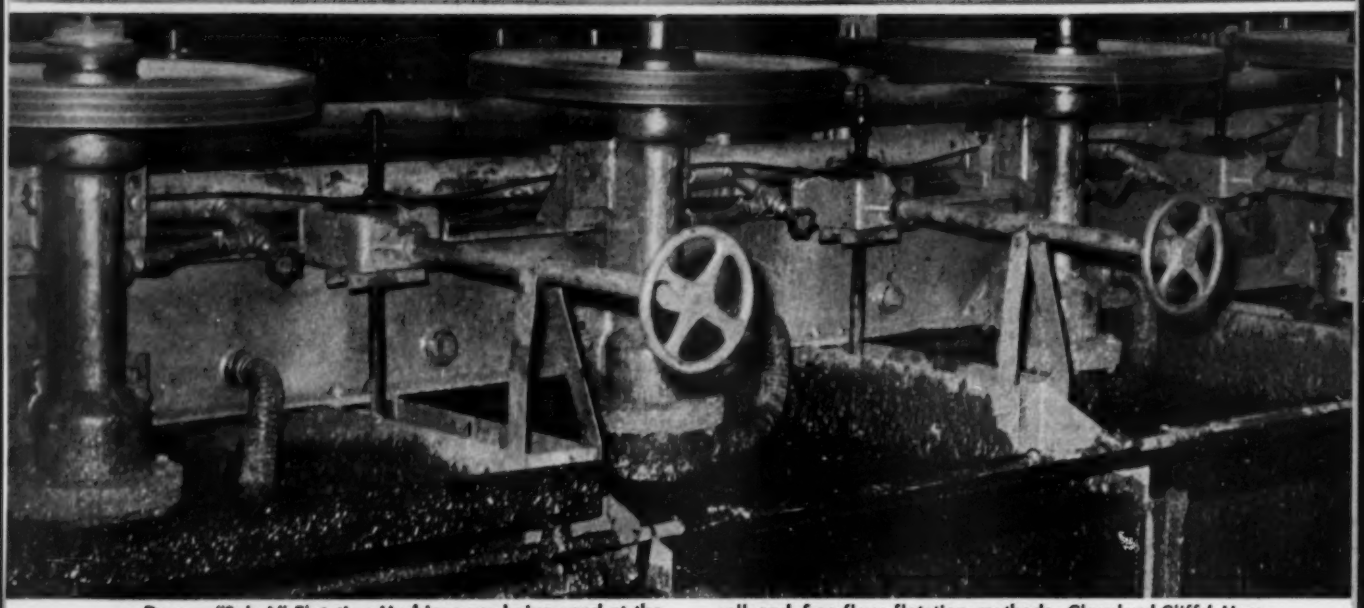
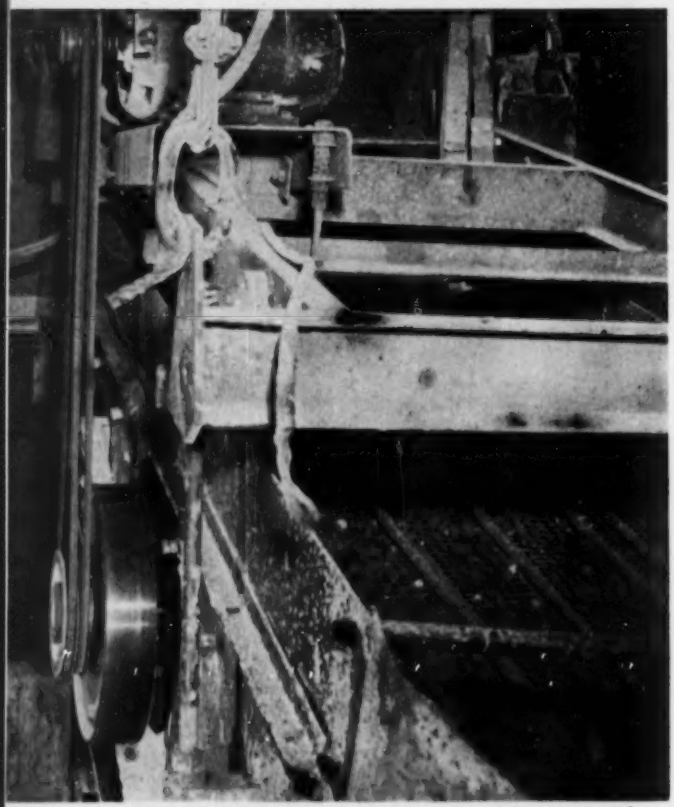
XP56-6

DENVER EQUIPMENT

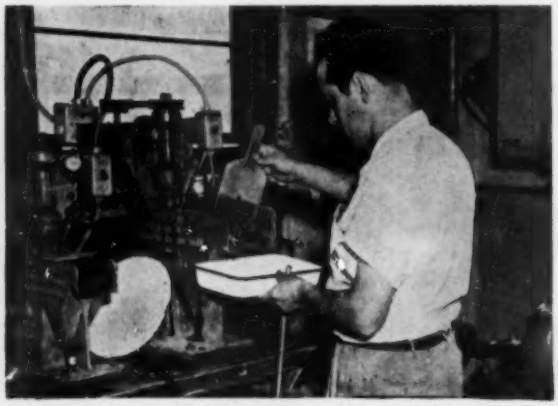


Denver "Sub-A" Flotation Machines are being used at the mill of the Humboldt Mining Co., operated by the Cleveland-Cliffs Iron Co., near Ishpeming, Mich., to float specular hematite iron ore. A fatty acid reagent is used. This photo shows a No. 24 (43x43) Denver "Sub-A" Flotation Machine in the Rougher Circuit. This bank of cells incorporates both cell-to-cell and free-flow flotation methods. Cleveland-Cliffs' Humboldt Mill is also using Denver "Sub-A" Flotation Machines for cleaners and recleaners. Additional information on how Denver "Sub-A" Flotation and Denver "Sub-A" Super Rougher Flotation Machines handle large tonnages is available. We want to help you solve your metallurgical problems. Item 311.



Denver-Dillon Screens give fast, clean separation without blinding. You get an even, smooth flow of material because of the patented "true-circle" action. Two bearing construction gives up to 50% savings in power. This mill photo shows the even distribution of the oversize—no blinding. Item 313.

60'x12' Denver Spiral Rake Thickener handling Fluorspar middlings from cleaner flotation cells. Patented Denver Spiral Rakes tend to eliminate overload due to their rapid removal of settled solids. Underflow metered with Denver Adjustable Stroke Diaphragm Pump. Item No. 315.

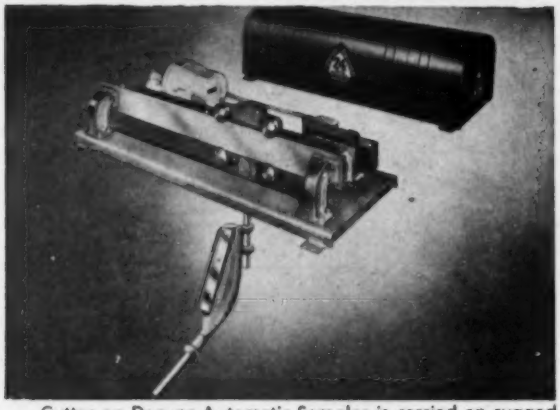


Denver "Sub-A" Laboratory Flotation Cells are used extensively by companies all over the world to check mill operation and maintain careful mill control. If you need lab equipment DECO has a complete stock—or you are welcome to use the Denver Laboratory Service for batch or continuous tests, or other problems. Item No. 317.

"The firm that makes its friends happier, healthier and wealthier"

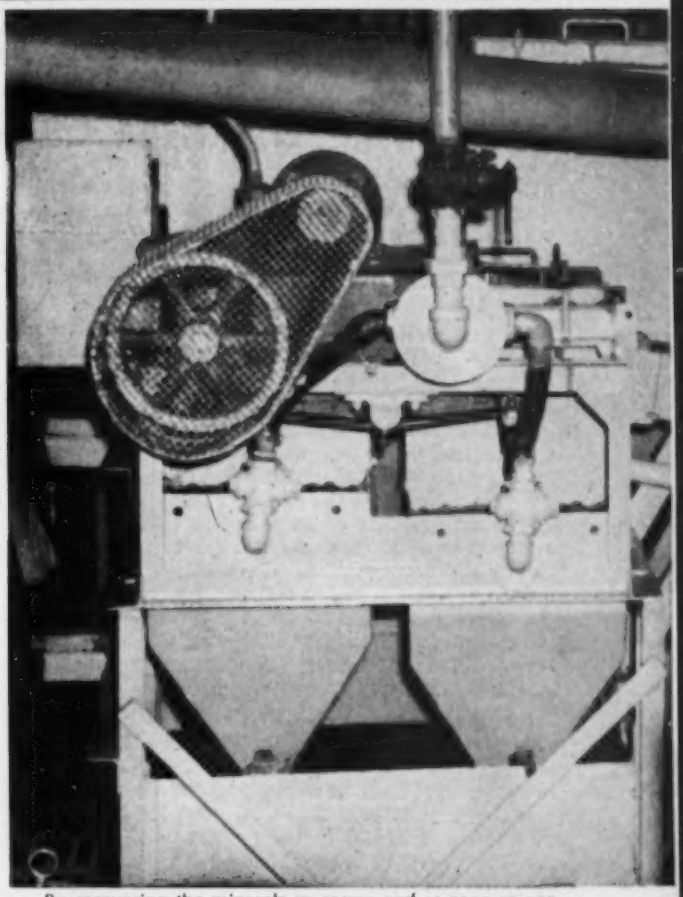
DENVER EQUIPMENT CO. 1400 17th St., Denver 17, Colorado

IN THE NEWS . . .

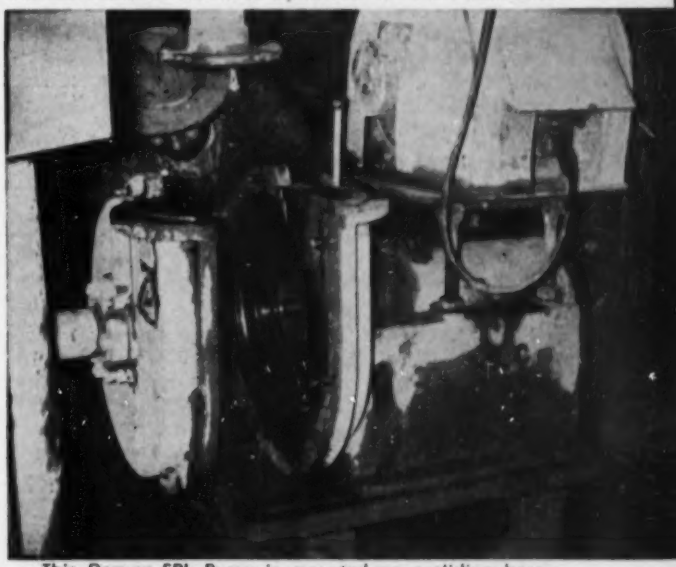


Cutter on Denver Automatic Sampler is carried on rugged ball bearing roller assembly, replacing old-style sliding contact. Rolling carriage allows the cutter to move through the stream of material to be sampled at a uniform speed. Frequency of sample cutting easily regulated. This gives accurate percentage samples, always. Item No. 316.

The 1955 Denver Wall Calendar and Mill Chart, and the 1955 Denver Annual Diary are being sent to everyone on our DECO Trefoil mailing list. If you would like to receive these two useful items, and DECO Trefoil, without cost to you, please send us your name immediately. Item No. 318. Sorry, but this offer can be extended only until Dec. 31, 1954.



By recovering the minerals as coarse and as soon as possible, Denver Selective Jigs avoid losses by over-grinding and sliming. Denver Jigs handle unclassified feed, make both fine and coarse concentrates. Up to 800 T/24 hrs. Item 314.



This Denver SRL Pump is mounted on a sliding base—making it easy to move pump back for quick inspection or parts replacement. Rubber - bonded - to - steel lining not only gives longer wearing part life, but also increases efficiency and lowers horsepower requirements. This 3"x3" Denver SRL Pump is used to pump cleaned zinc concentrates to cyanide circuit. Item No. 312.

WRITE TODAY, FOR ADDITIONAL INFORMATION . . . Please request additional information by the Item Number.

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EL PASO, 201 West 7th Street, Phone 3-8371.
SALT LAKE CITY, P. O. Box 705, Phone 84-8012.
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VANCOUVER, B. C., Credit Foncier Building, Phone Marine 4918.
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MEXICO, D.F., Avenida Juarez 14, Phone 21-1477.
JOHANNESBURG, Broadway & 9th, Bez Valley, Cable DECOJO, Ph. 22-7331

MERCURY MYSTERY—ATOMIC TIMETABLE—GYPSUM

WHAT happened to 50,000 flasks of mercury that seems to have just upped and disappeared? Along with the mystery of who's got the quicksilver is the rather unusual behavior of mercury in the market place. The flask price has been acting like a thermometer in mid-August—right now sellers are asking and getting around $330 per flask. One government contractor is almost certain to fall down on his end of the bargain because he can't buy mercury in the quantity he needs for his product, even though he's offering fantastic sums.

What would anyone do with 50,000 flasks? The immediate reaction to such a question is conditioned by the times. Atomic energy leaps forward as a logical place for the use of so much of the metal. Yet, consider this—the most obvious place would be vacuum pumps or in a reactor plant utilizing mercury as a heat transfer device. But, despite the magnitude of a vacuum pump setup, little or no mercury is lost in the operation of such a plant. In the case of mercury's use as a heat transfer medium, it is not the best available, according to present data. Mercury has a tendency to pick up radioactivity and thus becomes rather *hot* to handle.

Newspapers throughout the U.S. carried headlines several months ago concerning the U. S. Government's intention to encourage mercury mining in this country. General Services Administration was going to buy 200,000 flasks of domestic and Mexican quicksilver during the next three and a half years at a price no higher than $225 per flask. Even then, the going price for mercury was much steeper. Some exploration contracts have been offered in line with mercury's position on the critical minerals list. A few small mines have announced or are going ahead with plans to reopen. The New Idria mine in California is planning to start up operations. However, one large mercury mining outfit has announced that it has no intentions of reopening West Coast mines unless something more concrete is forthcoming. This in the face of a skyrocketing market.

A financial reporter for one of the nation's most respected newspapers says that he was told about the coming mercury boom last April. He didn't believe the story. Now he's wondering about the rest of the tale. His informant stated that the boom would be shortlived, reaching its peak somewhere around this winter and then quietly folding up.

There is one strong possibility in the search for an answer. Two countries, Italy and Spain, produce the greatest part of the world's mercury. Two others, the U.S. and Soviet Russia, may for the sake of argument be considered the biggest buyers. The mercury market place is confined to a relatively small space—with well-defined buyers and sellers. Suppose for the moment that Russia and the U. S., for vaguely defined reasons, have been going about buying mercury just so the other can't get its hands on as much as it wants. This would involve a certain amount of bidding. Could it be that the U. S. and the Russians are running the price up? Could it also be that they are both involved in a fantastic but short-lived shopping spree? If true, the answer to the skyhooked mercury price is apparent. (R.A.B.)

ONLY one thing remains in doubt concerning the atomic energy industry—"its timetable." Speaking at the American Mining Congress in San Francisco, Jesse C. Johnson, director, Div. of Raw Materials, U. S. Atomic Energy Commission, pointed out that plants built to produce electric energy from atomic materials may produce power at costs ranging from 5 to 7 mills per kw-hr. That's the target of companies now engaged in development work. Mr. Johnson made a few rough estimates of the uranium required in the future.

"Assuming that the nuclear power industry will require 14,000 tons of uranium metal a year by 1975 or 1980, what is the outlook for meeting this requirement, and at what price? Fourteen thousand tons of uranium metal is roughly equivalent to 17,000 tons of uranium oxide, U_3O_8.

"On the basis of our present knowledge of Free World uranium sources, a production rate of 17,000 tons, or even more, should be available at a price of not more than $10 a pound. This would be the price in terms of today's dollar for uranium oxide in a high grade concentrate. Much of this production might be sold profitably at less than $10."

Mr. Johnson noted that sources of the type now supplying the defense program might support a production of 17,000 tons or more for many years, assuming new deposits are found to replace those mined out. Jumping beyond the immediate present, Mr. Johnson labeled low grade phosphate and shale deposits as sources of vast uranium reserves. By-product uranium production has already started in the phosphate industry and will grow in importance, Mr. Johnson says.

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THE U. S. gypsum industry is expanding in spite of its passive attitude toward fundamental product research, according to the first management audit ever made of an entire industry.

The survey was made by the American Institute of Management, applying standards formerly employed by the nonprofit organization in evaluating the management of single companies. Main reasons for growth, says the AIM, is the emphasis placed on product line expansion and intensive mechanization of its manufacturing processes, resulting in substantial cost reductions. Contrary to general business trend, the overall gypsum industry spends relatively little for basic product improvement and long-range planning.

Use of gypsum can be traced back to Egypt about 5000 years ago. Next to lumber, it is the most common material in the building trades. Plaster, wallboard, sheathing, and lath are all gypsum products. U. S. Gypsum Co., largest of the firms, has tripled sales since 1945. National Gypsum has boosted sales four times. Despite labor troubles, two other major producers, Certain-teed Products and the Celotex Corp. doubled sales in the same period, paralleling the manufacturing average. The AIM forecasts con-

tinued growth for the industry in the next decade, keeping pace with the booming building industry.

While research activities have been moderate, effective developments in prefabricated materials stimulated industry growth. Diversification into building material lines other than those made from gypsum has also helped. Nongypsum products include insulation, lime, asphalt roofing, asbestos cement, shingles and siding, acoustical material, paint, asbestos, fiber insulation board, and metal lath.

There is no conformation by companies to an industry pattern. Management is of uniformly high quality, according to the AIM, but varies from hyperconservative Sewell Avery, director of U. S. Gypsum, to expansion-minded Melvin H. Baker, chairman of National Gypsum.

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ST. MICHAEL'S College is set amidst a 123-acre campus within the city limits of Santa Fe, N. M. Not unlike other colleges in the U. S. today, it has trouble obtaining the money needed to keep up with a growing enrollment and expanding facilities. St. Michael's is run by the Brothers of the Christian Schools, a world-wide order of the Catholic Church devoted to teaching. Right now the Christian Brothers are going about the business of raising funds for their school through Petaca Mining Corp., of which they own 55 pct of the stock. Petaca intends to mine mica, columbium, tantalum, monazite, and uranium in New Mexico and Utah. Brother Benildus, member of the order since he was 19, is the moving spirit behind St. Michael's College Foundation's advent into the mining world.

The tax-exempt educational and charitable foundation acquired the New Mexico mining claims from businessmen and prospectors. The claims were turned into Petaca Mining Corp. in exchange for stock. Petaca is offering 900,000 shares, advertised as a speculation. Two shares of preferred and one share of common are being sold as a unit—at $3 per unit. Total issue is made up of 600,000 preferred at 50¢ par, and 300,000 at 10¢ par.

Petaca will pay no royalties on the claims, but the foundation is obliged to pay more than 25 pct of corporation income to the original claim holders. One of the first steps to be undertaken by Petaca is the construction of a $300,000 mica mill with a capacity of 300 tpd. Petaca can draw on 22 mining claims covering some 440 acres in New Mexico. Brother Benildus says that Petaca also owns a 50 pct interest in 77 uranium claims near the Utah-Colorado border. Petaca expects to put some $25,000 into exploration of its uranium claims.

Petaca's property was discovered and mined, according to local history, by the early Spaniards before the Pilgrims landed at Plymouth. They named the area Petaca. The company will be completely staffed with people outside of the Christian Brothers order. Petaca expects a sales volume of $1.5 million from mica alone.

The college was founded in 1859 as a grade school. Five Christian Brothers came from France at the request of Bishop Lamy, who had journeyed to the area five years earlier. In 1947 the school became a four-year liberal arts college, taking over Bruns General Hospital and its 48 buildings through purchase from the War Assets Administration.

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FALLING ground is the No. 1 killer in U. S. mines. Each year about 420 men are killed and 7500 more are injured because of it. In noncoal mines alone falls of ground deal out death and injury to about 1 in 50 miners annually. Starting Jan. 1, 1955, the National Safety Council will conduct an intensive campaign to cut the toll by at least 50 pct. John L. Boardman, Anaconda Mining bureau of safety chairman, and chairman of the campaign, sums the NSC attitude by saying, "Although we realize we are setting our sights high, we feel that this is the only way to attack a problem that results in death and injury to miners each year. Falling ground is and always has been the biggest accident problem in the entire mining industry."

The National Campaign to Reduce Falls of Ground Accidents in Mines is sponsored by the mining section of the NSC. All noncoal mines in the U. S. and Canada may participate in the campaign providing 25 or more employees work underground. Membership in the council's mining section is not required. Endorsement and cooperation has been forthcoming from the U. S. Bureau of Mines, AIME, American Mining Congress, state and provincial mining departments, mining associations, mining publications, and others.

There are three basic requirements for participation in the campaign: companies operating two or more mines must enroll each mine separately; each participating company must agree to provide simple reports of injuries resulting from falls of rock and ore for the year, or two or three years, prior to 1955, depending upon the period for which records are available; each participant must send in a report of accident experience at the end of the first six months of the campaign and a final report covering the last six months.

The campaign is scheduled to end Dec. 31, 1955. Nonmember companies may obtain council materials for conducting the campaign at member prices. A plan is to be suggested for the systematic use of materials and the carrying on of activities. Companies may select materials and activities most suited to their needs. The drive will utilize posters, employee training booklets, visual aids, banners, publications, and safety stunts.

A Certificate of Achievement will be awarded to firms that carry on four safety activities and accomplish at least one of the following: reduce injuries from falls of ground by 50 pct compared with previous records, or have a record of no chargeable injury because of ground falls during the entire campaign year.

M. A. Matzkin

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Assure maximum recovery, and minimum cost in the flotation of sulfide minerals.

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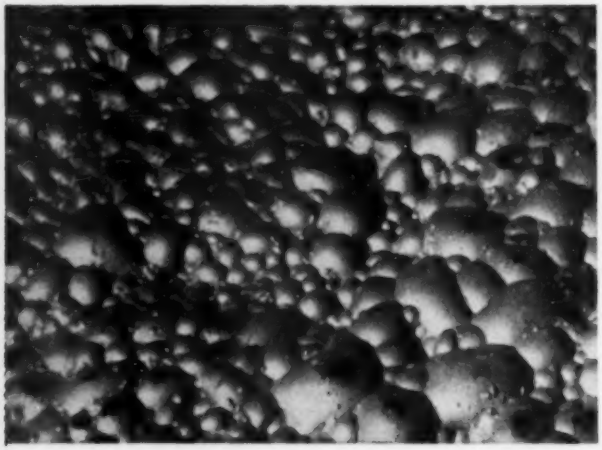

These Xanthates permit greater freedom in regulating frother and collector independently because they are reliable collectors but substantially nonfrothing.

For a quality frother, however, give Dowfroth® 250 a trial. It builds hard-working froth with reduced consumption.

To learn more about the quality flotation agents, Dow Xanthates and Dowfroth 250, write us for information and test samples. THE DOW CHEMICAL COMPANY, Midland, Michigan, Dept. OC 835I-1.

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Drift of Things

RECENTLY there have been several teams of European mining engineers touring the U.S. Sponsored by the Foreign Operations Administration, these teams have usually visited AIME headquarters before the tours, to exchange ideas. Reams of campaign oratory to the contrary notwithstanding, some things are taken too much for granted in this country, and it takes visitors from abroad like these FOA groups to drive home just how much certain of our conditions are taken for granted.

One group found interest in U. S. shortages of engineering students, and in the organization of Engineers Joint Council, and of EJC's Engineering Manpower Commission. Further lines of questions concerned engineer registration and licensing. Confusion was revealed by the visitors' queries on official governmental action and policy in manpower matters and it took quite a bit of emphasis to make clear the wholly voluntary aspect of manpower utilization characteristic of U.S. policy even in wartime.

Rather obvious that—but not to these men from Europe. Talk about engineering manpower led to questions about the U.S. industrial set up. Especially those who had not been to the U.S. before appeared to have an attitude of amazement that the U.S. gets anything done, rather than one of envy or respect for our institutions. Luck seemed to get a lot of credit for U. S. material successes. That may have changed by the end of the trip.

PERHAPS, and it is a small perhaps, where the U.S. must do its selling job in telling others of democracy is in convincing the skeptics that our results stem from something other than luck. That the apparently happy-go-lucky system really works best. Some of these gentlemen politely wondered that it worked at all, especially in wartime.

Respectfully leaving problems of psychological warfare to the brains of the Pentagon and Madison Avenue, there does seem a parallel that can be drawn for AIME. What puzzled these Europeans was that this country gets anything done at all considering individualism and industrial competition. Some ventured the idea that in some mysterious way a majority of the people get the same idea at the same time, and moving individually, but simultaneously, progress toward an objective.

Again we hesitate to draw homilies on national philosophy from small talk, but within the smaller realm of AIME activity that certainly seems the truth. To get anything done requires a majority of individuals, but still individuals, all pushing in the right direction. With such a feeling alive the leaders and doers get results, and without it they must do the job alone, or not at all.

Growth of interest within the Mining Branch is one example. It seems about time for all members of the branch to push in some one direction, and get some new projects moving. Being publication-minded several book projects occur but there are plenty of other items for an agenda—such as an all-Mining Branch fall or regional meeting for next year. To paraphrase the cliché used in drawing room comedies, "Anyone for tennis?" one might say, "Anyone for.....? and let the members fill in the rest.

IN an era of change it's reassuring to see old familiar faces. In this case the face is the cover of the 163rd edition of *The Old Farmer's Almanac(k)*, continuously published since 1792. The entry for January 22, "1st Atomic Sub Launched 1954," proves that the Almanac moves with the times. But this item from the atomic age is cheek by jowl with the one for January 27: "Boston's 1st Fire Eng. 1679/Mercury in Ascending Node/Moon on Eq./9.7 10.2". All in all a neat package of information for the 27th, the latter two figures being heights of high water at Boston. At least one can count on something being in the ascent that day, and *these* days that is something. For the period January 21 to 31, the weather is described: "Thaws a while/but don't smile/it storms begorrah/today and tomorrow." We've seen geologists and engineers who weren't as explicit in prediction. And compared with this, the Official Forecast is child's play—it covers only the next 24 hr.

The old book is stuffed with wonderful bits of data and minutae. Principal holidays for 1955 are listed, such as Mecklenburg Day, May 21, (North Carolina), and Repudiation Day, November 23, (Maryland). That last is handy to have around. All sorts of things one could repudiate. Need conversation starters? There are things like the average seasonal snowfall at Tamarack, Calif. (It is 351 in.)

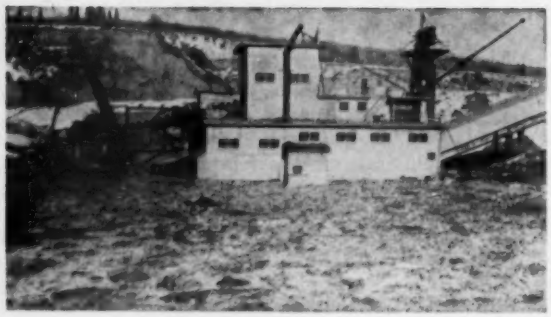
Seeing the 1955 almanac is fair warning that 1954 is drawing to a close—and with it, completion of the round of Mining Branch fall meetings. This month *ME* covers the MBD fall meeting in San Francisco, and in the December issue look for full coverage of the Pittsburgh, Salt Lake City, and Lake Placid get-togethers.

In a file folder plainly marked "Drift" there accumulates a collection of flotsam. No jetsam, thank you. Here it is . . . The almost 49th state, Franklin, was called to everyone's attention in a booklet "The Lost State" from the Atlantic Companies, who incidentally sell insurance. The idea blew up in 1788, so it isn't a burning political issue at present we hasten to assure you . . . Gold prospectors can get an early start with an "educational hobby kit" from the Gemological Institute of America. Contents: geologist's hammer and chisel, instructions, lens, etc. . . . The AEC has released some information on a low tonnage use for iron ore. In this case it becomes, "magnetite iron ore concrete . . . aggregate for a dense shielding concrete . . . blocks used in hot cells" . . . According to the *Vermiculite News* gamblers like peace and quiet, too, when they're at work. That's why a Wilders, Ky., gambling hideaway has treated its ceilings with vermiculite acoustical plastic. Patrons can concentrate on losing their money in hushed comfort. We wonder if the gamblers, reportedly in Wilders, would just prefer a hush about it all . . . Safety engineers have their troubles too. A routine production department fire was a simple case of a man using arc welding equipment igniting a tank containing crude oil. What was he welding onto the tank? A large metal "No Smoking" sign.—R.A.B.

See you in Chicago!

Arctic Gold Dredging

by Patrick H. O'Neill



Pond ice in early spring typifies Alaskan operations.

FUNDAMENTALLY a dredge designed for operating under arctic conditions and particularly when the temperature is below freezing is not greatly different than one for use in more moderate climates. Because of the accumulation of ice it is necessary to design suspension units to carry an increased weight when ice forms on the digging ladder or stacker. Winch room and house must be made with double walls and insulated. The dredge must be heated either by steam or by circulating warm air electrically heated. The stacker must be covered and heated to avoid having the drive units frozen tight. Actually the worst trouble that might develop in a stacker, because of cold weather, is the freezing of spilled material and ice accumulation on the head pulley and the underside of the belt. For this reason, stackers are completely enclosed and the pan kept open to get rid of any spilled material and to prevent overloading. Since the stacker is the most vulnerable place on a dredge in cold weather, heat is turned on there long before it is necessary any other place on the dredge. The principal points of interest in connection with dredging, under subfreezing conditions, concern the preparation of the ground being dredged and the frozen ponds.

In the interior of Alaska, and also in the Yukon territory of Canada, most of the gold bearing gravel deposits are permanently frozen and they are also covered by variable thicknesses of fine grained sediment, locally called muck. The better pay areas in this frozen ground have generally been worked by the earlier miners who sank shafts to bedrock in the frozen ground, then drifted in four directions and dug out the areas between the drifts as they retreated toward the shaft. They thawed only the area they were excavating, doing so with steam from wood-fired boilers, and hoisted pay gravel to the surface for subsequent washing.

Frozen condition of the gravel and muck enabled early miners to recover the main concentration of values on bedrock, at depths that often exceeded 200 ft below the surface. Before dredging could be carried on successfully, however, methods of economically removing the muck and thawing the gravel had to be developed.

PATRICK H. O'NEILL is now Assistant Vice President, South American Gold & Platinum Co., New York.

Muck overburden varies in thickness up to 150 ft and is easily transported with water since a large percentage of the muck particles will pass a 200 mesh screen. In the large-scale operations, stripping is done with hydraulic giants placed so all parts of the area to be stripped can be reached by water from the giants. Working radius of a giant varies from 100 ft with water pressure of 50 psi to 225 ft with water pressure of 150 psi. Freshly exposed frozen muck generally thaws about 1 in. during the first 2½ hr, then the rate of thaw decreases due to the insulating effect of the thawed material. A typical thawing rate would be about 3 in. after 8 hr exposure but only 4 in. after 24 hr exposure. On a well-planned stripping operation approxi-

Cribbed shaft used in early drift mining still stands in muck stripping area, now being dredged. Pioneer placers operated underground from shafts like these using a retreating method of mining. Costs necessitated mining high grade gravel.

Ice formation on lower ladder suspension blocks is one hazard of cold weather operation. Headaches like this sharply curtail operating season.

mately 12 giants are set up for each available stream of water; each giant is operated 2 or 3 times in a 24 hr work day and only long enough at a time to wash off the accumulated thawed material. The water can carry from 10 to 20 pct by weight of suspended solids and drains are provided to carry the material to the nearest stream or river where it is further diluted and carried away. No debris problem occurs because the major rivers originate in active glaciers and the percentage of silt introduced by stripping operations is small in comparison to that introduced by the glaciers. Actual hydraulic removal of muck is, of course, limited to the short summer season when water is available and usually does not exceed five months. Water for stripping, depending on supply and available operating head, is provided by ditches or pumping plants or from a combination of both these sources.

Gravels were thawed with steam during early dredging in the interior but this way was very costly and only high grade gravels could stand the expense. In the 1920's a cold water thawing method was developed, thus expanding the potential dredging areas by greatly reducing thawing costs. In most Alaskan dredging areas the frozen ground temperature is about 30°F. Water at summer temperatures, 45° to 50°F, is pumped into the ground through pipes that extend to bedrock. As the water percolates back to the surface the frozen ground absorbs heat from the water and thawing occurs. As the ground thaws, it settles away from the frozen area and becomes denser so the line of least resistance for the water is along the frozen area and thawing progresses outward in a circular area from each pipe. The time required for thawing a given area depends upon the character of ground, water temperatures and pipe spacing. A typical rate is 1½ days per ft of depth when the pipes are spaced to form 16-ft equilateral triangles and 2½ days per ft of depth with 32-ft triangular spacing. Closer spacing is generally used in ground shallower than 45 ft where ¾-in. pipes are driven by hand or with recently developed driving machines. The wider spacing, 32 ft, is used in ground over 45 ft deep where holes are drilled to bedrock with standard churn drills and 1¼ or 1½-in. pipes are placed in the drilled holes. The subsidence that occurs during the thawing results from removal of the ice in the ground and to some extent from settling into the old drifts of the earlier miners. Artificially thawed ground is therefore loose, digs easily, and caves or sloughs ahead of the digging to the extent that it is frequently not necessary to raise the bucket line above water level when stepping ahead.

Bank Stabilization by Freezing

Bank stabilization can be an important consideration for the safety of a dredge. In subarctic climates it is possible to employ the low winter temperatures to freeze pillars for this purpose as shown by one such undertaking near Fairbanks. This produced a paradoxical situation since a short distance away crews were engaged during the summer in thawing the ground so it could be dredged and here the undertaking was to freeze naturally thawed ground to keep it out of the dredge pond. The dredge was operating in deep ground close to a high bank of silt overburden. The water table was near the surface and there was a clay stratum just above bedrock. Fifteen pillars each 30-ft diam and 135 ft deep were frozen to bedrock near where the dredge was scheduled to reach the high bank. The pillars, on about 40-ft centers, were frozen during the win-

LEFT: Installation of truss work proceeds on pontoon hull. Upper bank is face of muck stripping and lower bank is face of barren gravel stripping. RIGHT: The completed dredge is ready for the season. Muck stripping shows in foreground, and dragline is stripping gravel in background.

LEFT: Ice is usually cut with steam, but here a power saw is being tested. Steam cutter is in use by man at right. The pond level has dropped leaving a ridge at the sand bar. RIGHT: Cutting ice over large area with steam cutters, men breaking cakes loose for removal.

ter by circulating a low-inflammable gasoline mixture through a battery of fan-cooled tractor radiators, then through a system of 4-in. closed-end pipes extending to bedrock in drilled holes. These pipes had smaller diameter thin-walled drop pipes in them to deliver the refrigerant to the bottom from where it returned to the surface in the space between pipes. The refrigerating pipes, which were arranged in trianguiar pattern at about 14-ft centers, formed small frozen pillars that eventually coalesced to form a major pillar. The purpose in leaving thawed spaces between frozen pillars was to prevent the accumulation of a hydrostatic head behind a frozen barrier. The freezing was done in one winter. To prevent thawing, by circulating ground water during the summer, pumping of the refrigerant was continued and cooling to a temperature below freezing was accomplished by electrically operated refrigerators.

This 2-ton ice cake rides the highline to the pile on the bank. Trolley uses highline, and backhaul is by smaller jitney line.

Dredges cannot operate the year around in Alaska or the Yukon because of the cold weather. Ice begins to form on the dredge ponds in late September or early October but usually does not slow up operations appreciably until early November, when the dredges have difficulty in moving from one cut to another. This applies especially to the deeper digging dredges that do not move so often as shallow digging dredges do. An ample water supply improves the chances of late season operation but a two or three day spell of 15° below zero or colder weather can force a shutdown, for the ice forms faster than the dredge can break it. Operating costs are greater in cold weather, because of heating costs and the necessity of frequently removing ice that forms on the digging ladder and hull. Locomotive type boilers are generally used for heating and to provide steam for heating water to wash off ice and frozen dirt.

Pond Ice Removal and Steam Thawing Seasonal Frost

In most parts of interior Alaska the average temperature is usually above freezing by the middle of April, but the ice and the winter frost are not thawed until the middle of May. The dredging season can be extended as much as two months in the spring by removing the pond ice and thawing the winter frost in the ground.

The pond ice varies from 2.5 to 5.0 ft in thickness. A common method of cutting the ice is to use a steam cutter made of ¾-in. black pipe joined together in rectangular pattern 2 ft wide by 6 ft high with a steam inlet at the top. The bottom cross-

Stream thawing winter frost ahead of dredge is one of spring start-up jobs, may also be done in fall to extend operating season as much as possible.

Ice cut as shown in photos at left is choked and hoisted to clear pond area.

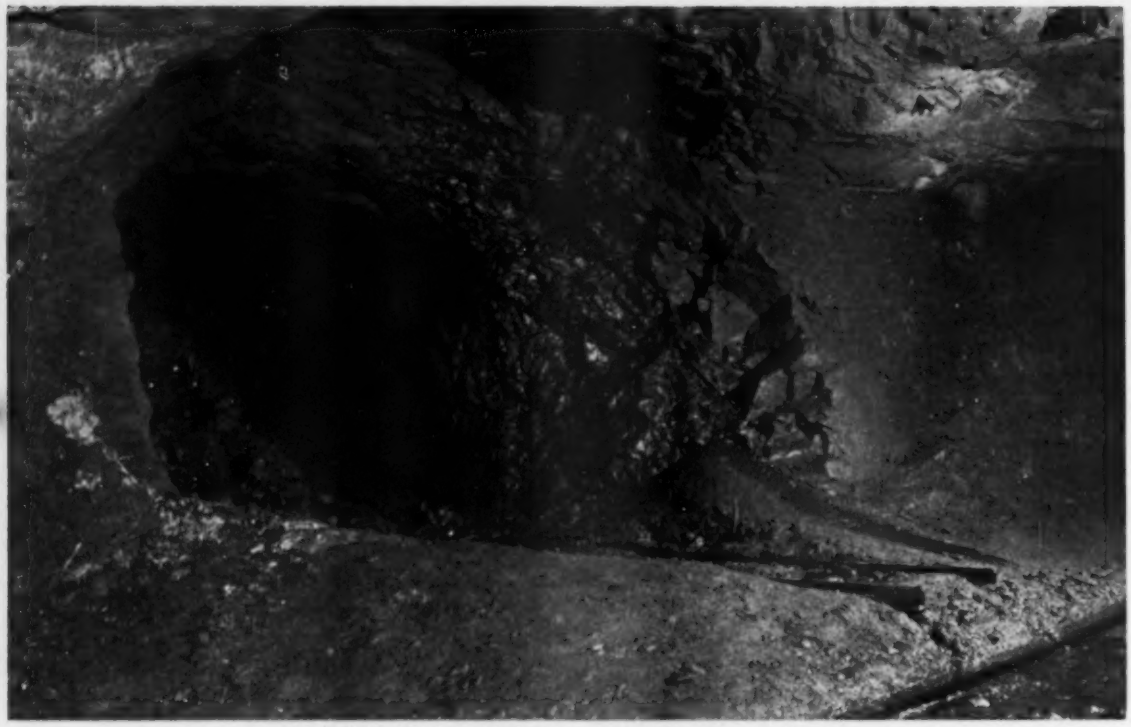
Cake of ice cut from pond area traveling to pile on bank. Start-up poses unusual materials handling problems.

piece is drilled with small closely spaced holes from which steam issues to cut the ice. The cutting rate with such equipment is generally about 11 in. per min with 100 psi steam pressure at the boiler. The ice is usually cut into cakes 3 to 3½ ft wide by 6 to 7 ft long. The size to be cut depends on the thickness of the ice and the capacity of the hoisting equipment. The cakes may vary in weight from 2500 lb to as much as 5600 lb. Anywhere from 2000 to 10,000 cakes of ice are removed at a single dredge at the start of a season. From one to three weeks is ordinarily required for ice removal in the spring.

Steam thawing of the surface frost immediately ahead of the dredge is usually carried on concurrently with ice removal. The winter frost penetrates anywhere from 3 to 9 ft but the average depth is about 4 or 5 ft. The depth of the penetration depends on the amount of snow covering, the type of material, winter temperatures and ground water level. Such other factors as mineralogical content, size of the particles, and degree of compaction of the soil undoubtedly have some effect on the depth of winter frost, but little research has been conducted along this line. The frost is thawed by steam delivered to the ground through a steam point. The points most widely used are similar to the steam points developed by the early drift miners in Alaska and the Yukon. They are made of ¾-in. double extra strong pipe with a forged head and a square tapering point. The steam enters a nipple on the side of the forged head. There is also a hole through the forged head to insert a small bar that is used to twist the point while pounding on the head. The boiler location depends upon the terrain and availability of water but should not be over 300 ft away from the face of the dredge cut. The boiler should be of sufficient size and be operated at sufficient pressure to maintain at least 20 psi at the end of the line or the points will freeze in cold weather. Insulation of pipe lines has obvious advantage but because of rough usage and frequent pipe line moves the use of insulation is ordinarily limited. Steam is delivered to the points through hoses attached to crossheads provided at 10-ft intervals along the header lines. The points are usually driven to a little above bottom of frost on 6-ft triangular spacing. With this spacing there is generally sufficient thawing in 8 hr to permit digging with a minimum of large frost chunks. The thawing program is planned to stay ahead of dredging and ground is thawed to where the dredge will be about the time when normal thaw will have penetrated at least 2 ft. It does not pay to thaw the frost remaining after that time.

THE FINALE: Dredge tailings on Goldstream, with brush already growing on older tailings.

Because of the high cost of steam thawing, ice removal, and other expenses incurred in early spring opening, a considerable extension of the dredging season must be realized to justify such work. In addition to the added costs of cold weather operation there are other disadvantages to seasonal operation such as lack of steady work for employees and resultant turnover. On the other hand, major repairs to the dredges can be accomplished during winter shutdown so that a high percentage of operating time can be attained during the operating season. Dredges can be moved to new locations, in large sections on sleds during the winter, saving much in dismantling and erection costs over movement by truck or rail. The necessary ground preparation and cold weather operation in general add considerably to the cost of dredging and unit dredging costs in the Arctic are two to three times those pertaining to year-round dredging in warmer climates.



View of drawhole from side shows track raised above main level track.

Vermont Copper Uses Mucking Machines For Stope Drawing

by Clinton L. Miller

MINING operations have been carried on in the immediate area of the Elizabeth mine during periods of high copper prices since 1793, when copperas and vermillion were first won from the outcrop. Present operations started in 1942 as a *war baby*. Since then production has totaled 1.7 million tons of 1.8 pct Cu ore.

Elizabeth mining operations are on a pyrrhotite-chalcopyrite orebody contained in a gently plunging amphibolite syncline. The oreshoot varies in width from 10 to 40 ft, and in the area under consideration is about 150 ft in vertical extent, and more than 1000 ft long. Surface pits or cuts have been worked in three areas along the outcrop, with one pit just completed and a second in full production. Present production is 24,000 tons per month, of which 6000 to 7000 tons is trucked from the open pit.

Two Extraction Methods

Both conventional scrams and mucking machine drawholes are used. Scrams are installed wherever the bottom of the orebody lies more than a few feet above the drift. The scram is driven as close as possible to the bottom of the oreshoot. Hoe type scrapers with hoists of 15, 20, 35, and 45 hp move ore from drawholes to transfer chutes. The mucking machine drawpoints are now installed wherever the oreshoot intersects the main level. Planning of either layout is prepared on a basic premise: to extract a minimum of waste and a maximum of ore with as little timbering as possible.

Late in 1950 the decision was made to install mucking machine drawpoints on the North 975 level of the Elizabeth mine rather than the timber chutes previously used for ore extraction from shrinkage and sublevel stopes. The orebody was thought to be a comparatively shallow and wide lense of high grade copper, and the operation was badly in need of the ore this portion of the shoot contained.

Mucking machine drawpoints offered maximum possible extraction of a small orebody which could not bear, and did not require, the expense of a timbering method. With this method stopes would be in production in a comparatively short time and at low preparation cost.

A secondary advantage of this method was the dewatering of the broken ore. Scramming operations under the wet orebodies had produced a constantly wet ore which hindered screening in the crushing plant. In addition, wet muck in chutes represented a serious hazard while loading trains. The use of sill-level drawholes allowed broken ore to partially dry out, and alleviated the screening bottleneck.

Large boulders in the drawholes made it necessary to blast during the shift with consequent problems in ventilation, mucking machine protection,

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and safety. In addition the stoppages slowed loading, and interrupted haulage on the main line.

Secondary Breakage

Proper fragmentation is basic in simplifying almost all of the inherent problems in tramming operations. The solution to this problem was to first determine the approximate drill hole spacing at which maximum fragmentation occurred, with the minimum powder and drill footage. This general spacing was experimentally determined to be about 3 ft in the sulphide ore. A 3-ft spacing rule was set up for the stope benches and breasts and the stoping crews and foremen and the entire drilling crew and bosses were instructed to follow this pattern.

Results of this campaign have been good in both the mucking machine drawholes and in the scrams. The oversize boulder has not been completely eliminated, but it is no longer commonplace. Consider the simplicity of haulage in a mine if the stope muck could be broken like development ore!

Maintenance

The maintenance of mucking machines was critical. The muckers required not only periodic major overhauls but constant minor repairs. The worst problem was damage to buckets resulting from frequent blasting with too heavy or injudiciously placed charges in the drawholes. The key to maintaining high production with this method was to keep the muckers in constant use, but not at the expense of neglected minor problems which sooner or later became serious. An inspection and maintenance system was set up with a mechanic on each tramming shift whose duties included servicing and checking each unit, *and recording the inspection.*

Air hoses, which were constantly being cut by the rolling equipment, were a minor problem. A steel-mesh reinforced hose was first used on the mucking machines with excellent results. Steel reinforced air and water hose is now used throughout the operation, and hose breakage has decreased materially.

Cars are 40-cu ft, rocker-bottom type. Most of the loading is over the side of the car with the mucking machine track 7 in. above the main level rail. This combination plus the policy of loading the largest possible boulders resulted in serious chassis failures.

The original cars were reinforced in the shop, where plans for a modified design were prepared. A more rugged design for the chassis ends was worked out with the manufacturer and this now stands up well in service.

The key to continuous operations proved to be a planned *and recorded* maintenance program, plus heavier hoses and cars, with top job priority in the shop going to a mucking machine repair.

Ventilation was first provided by coursing air through existing open stopes above the drift. This provided about 16,000 cfm, but as the drift advanced 1000 to 1500 ahead of the upper workings, a surface connection was indicated. Raising was not possible

Vermont Copper Found—

During the evolution of the mucking machine drawpoint method several interrelated problems were encountered:

1. Fragmentation
2. Maintenance
3. Ventilation
4. Drainage
5. Train scheduling
6. Drawhole layout

But attention to these six points made drawhole mucking work from single track layout and led to—

Three Big Advantages

Improved extraction

Cheaper and faster stope preparation

Dewatered stope muck

Closeup of mucker, drawhole, and tram shows necessity for proper cleanup, careful scheduling, and proper drawhole design. Eimco muckers receive careful maintenance on exact schedule. Author says key to continuous operation was planned and recorded maintenance program. Cars used at first proved too light, and original cars are now reinforced, with newer, more rugged design in use as well.

because a 90-ft mantle of water-soaked glacial gravel overlies the bedrock.

The problem was solved by churn drilling 12-in. holes from the surface in the hanging wall to the 975 level and casing through the gravel to solid rock. A short crosscut connected the churn drill hole with the main drift. Fans were installed to blow air to the level, or exhaust it as the season required. The churn drill holes proved a successful aid to ventilation and in addition served as power cable inlets from the surface. Two churn drill holes are now in use, with a third planned.

Ventilation in overlying stopes was improved by the installation of a fan so located that it drew fresh air over the top of the stope block resulting in a down pressure at the drawholes which forced the blasting smoke and gases out the main drift.

Fuse smoke was eliminated by changing to electric blasting, which also cut down the time lost per blast, and cut exposure time in the drawhole.

Control of spillage in this type of operation is important since the fines will sooner or later work their way into the ditch, which overflows and floods the track. A ditching program is constantly carried on, coupled with a sales campaign on housekeeping, from the sill up. The initial work involved in maintaining an adequate ditch was difficult, dirty, and discouraging, but once the work was completed, the problem of keeping it clean was easier. Each tramming foreman is responsible for cleaning up any spillage occurring on his shift. It is evident that a clean, well-ditched sill not only increases production and simplifies maintenance, but also results in a vastly improved attitude on housekeeping in all phases of mining.

The scheduling of the mucking operations was vital because of the necessity of producing as much as possible from this one drift, at first because of the higher ore grade, and later, as the older sections of the mine were worked out, for constantly increasing tonnages. Since two, and later, three trains were working on a single track, with only one switching point, the close coordination of all units was imperative. The present plan utilizes a 39-hp diesel motor which shuttles from a midway switching station located some 2000 ft from the shaft, to the dump and return, and two battery-powered locomotives which serve the drawpoints and haul from more remote development headings.

Scheduling

Scheduling is based on keeping each mucker working as much of the time as possible by:

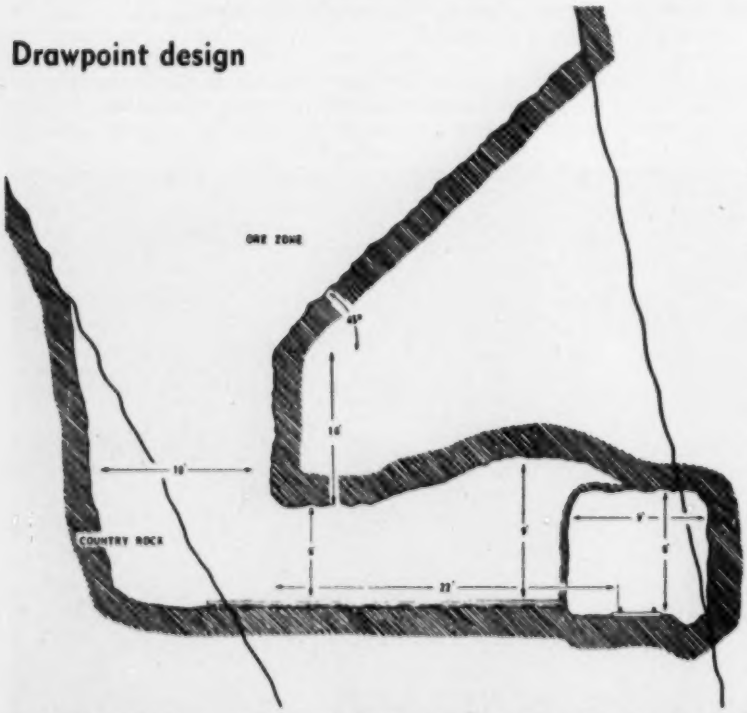
1. Working any given drawhole for as many shifts as possible before moving.
2. Performing all possible maintenance chores without moving the mucking machine.
3. Using blasters to work the drawholes down and break up the large boulders while the motorman and the mucking machine operator are switching the string. These men also prepare other drawholes for the following shifts.

The design and cutting-out of the drawholes has been experimented with at length. The accepted desirable design is a drawhole cut out according to the accompanying sketch.

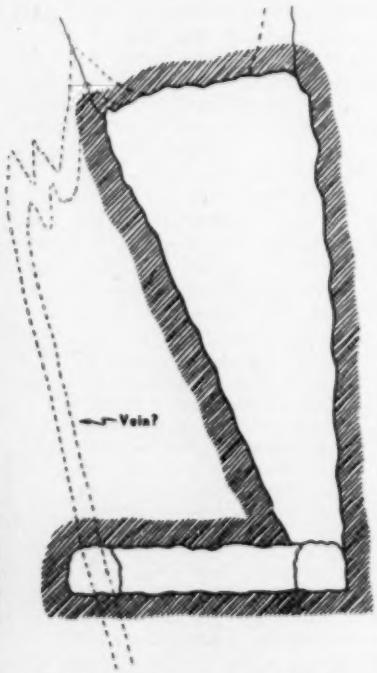
Mucking machines are moved by means of a *dolly* or transfer car, from one drawhole to the next, which accounts for the 7-in. step from the main level to the drawhole track.

Experimental work is being done with drawholes in which the rail is at the same elevation as the main drift rail. In this type layout the mucker is swung 90° on a steel slick-sheet. It has the advantage of not requiring a dolly for moving, but the serious disadvantages of requiring several men to

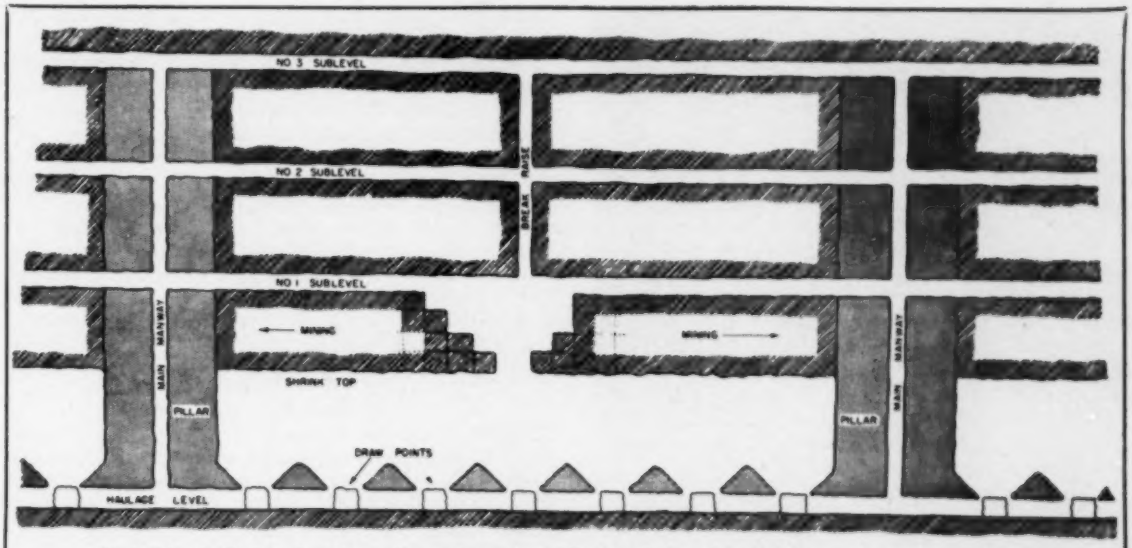
Drawpoint design



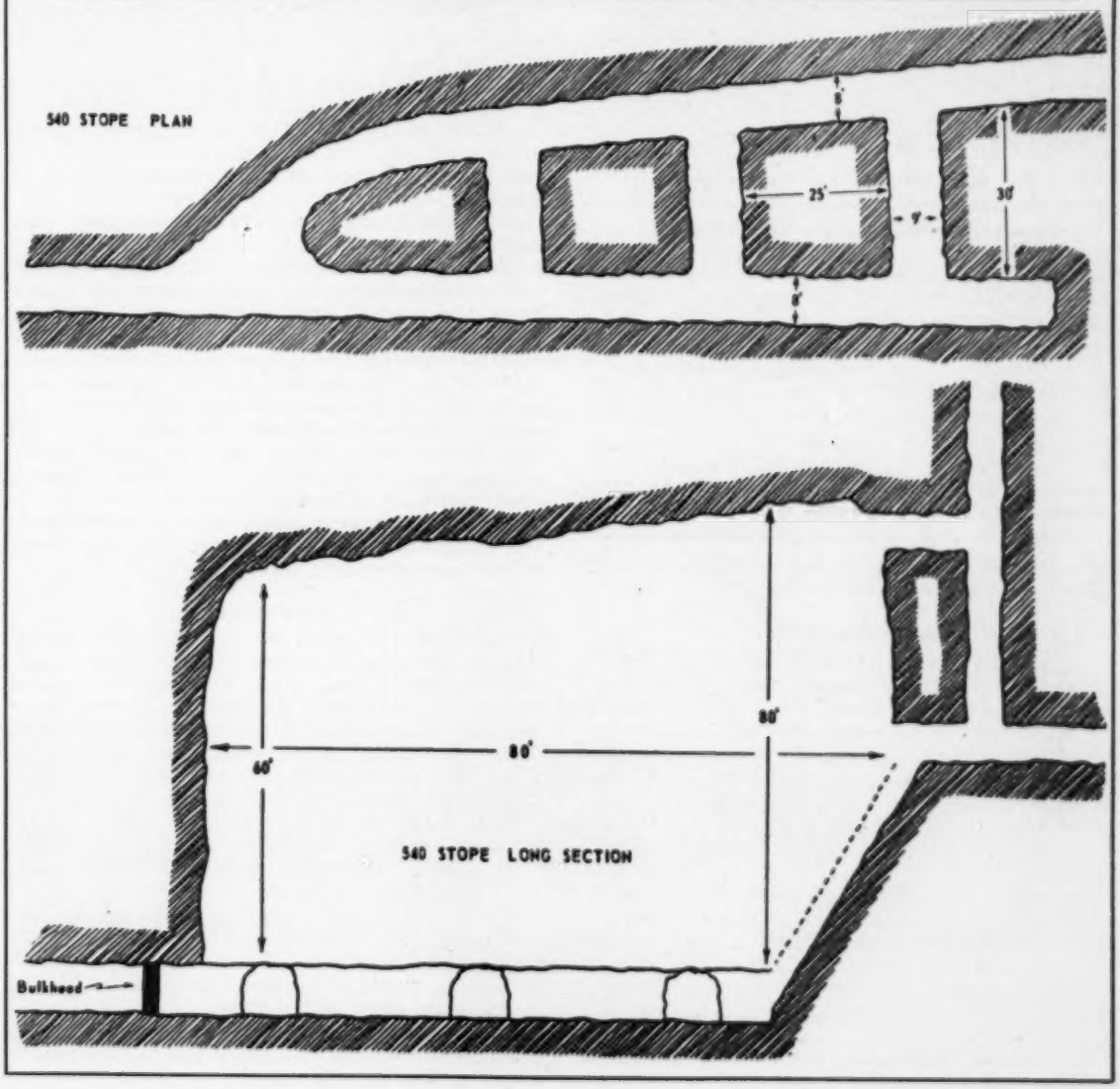
Desirable design for drawhole cutout is this layout. Drawpoint track is 7 in. higher than haulage track. Haulage drift at right of view is 9x8 ft and the crosscut is 8 ft wide.



Special mining layout necessary to handle split in vein is illustrated by 540 stope. Section of 540 is shown above; plan and section are at right.



Typical longitudinal section, above. Plan and section of 540 stope, below, show special stope layout for split vein.



Head-on view of drawhole shows brow design and raised track level that is 7 in. above main track.

swing the machine, and the harm done to the mucking machine while swinging on the horizontal bearing race.

One especially interesting operation was used in mining a split in the vein. The drifts were extended parallel to one another with approximately 30 ft of rock horse between the vein zones. Appropriately spaced crosscuts were driven to the drifted-out ore section, and stoping begun at once. This method of using drifting for the undercut, brought a shrinkage stope into production without the time-consuming delays and expenses involved in a standard cone-

Operating Data January to September 1953

Tons handled by mucking machines	86,387
Mucking machine shifts	1,403
Tons per mucking machine shift	61.6
Maintenance Costs	
Materials, parts, and lubricants	$2,741.26
Labor	2,158.38
Total	$4,899.64
Maintenance cost per ton handled by mucking machines	$0.057
Maintenance cost per mucking machine shift	$3.49

raise undercutting program, and allowed practically complete extraction of the orebody.

Undesirable Patterns

1. Drifting out the entire width of the orebody followed by driving drawhole raises up the footwall. This type of layout makes it necessary to muck from the side, off a switch, with the inherent disadvantages of partially loaded cars, and the work represented in switching each car. In addition it was necessary to erect timber bulkheads to prevent muck surges from blocking the main drift. These are expensive to install and require constant maintenance.

An additional disadvantage is the safety problem involved in an operation where the mucking machine operator is working either between the drawhole and the mucker or the mucker and a bulkhead. In each instance a quick surge of muck may cause a serious injury to the operator, a damaged mucking machine, or both.

2. Mucking from side-switches into drawholes at right angles to the track. This operation has the same load limit and switching disadvantages as the previous instance, with the added drawback of mucking on a curve, with resultant spillage, and slower loading. One advantageous aspect of both these layouts is the fact that a dolly for transporting the machine from hole to hole is not required.

3. Drawholes in which the brow is too close to the main track—either from poor layout, improper drilling, or erosion of weak or faulted zones. The muck surges either cover the track so that the mucker cannot be driven onto the rails, or bury the machine so that it must be shoveled out. When blasting is necessary in these short-browed drawholes, it is generally so close to the mucker that the possibility of damage is great.

Present Operations

A 39-hp diesel locomotive hauls the loaded cars from a switch located at the first of the drift-level drawpoints to the station grizzly and return. The experience with the diesel motor has been excellent: more power and less maintenance. This shuttle haul is about 2000 ft one-way and is increasing.

Three mucking machines are kept in operation in the areas above the switch, two in drawholes and a third in drift headings. A wide group of drawhole types are still in production, but the new ones are all cut out to the patterns illustrated above.

A 1½-ton Mancha motor and 6-car string of 40-cu ft cars works the drawpoints close to the switch, while a 3-ton model plus a 12-car string services the drawpoints and headings farther north. In each case the motorman's helper is also the mucking machine operator.

A team of two blasters has proven valuable. They move from one mucker to the other working down hang-ups and blasting large boulders when necessary and keep temporarily inactive drawholes blasted down and ready to be pulled by the next shift. Blasting is done, if possible, while the mucking machine operator and the motorman are switching the strings. The use of a blasting team has very definitely kept the machines loading a greater percentage of the shift than when the mucker crews did their own blasting. One advantage is the possibility of staggering lunch periods so that the mucking crews can work to the best efficiency.

The average haul is 3800 ft from the drawhole to the station grizzly, and is necessarily becoming greater as stoping progresses to the north.

Conclusions

It has proven feasible and, under specific circumstances, desirable to operate drift-level drawpoints from a singleline main track. A number of problems were encountered in working out this method, many of which were anticipated, and several that were not.

Major disadvantage of this type of tramming system is the inevitable drift tie-ups, which do occur, despite the most careful planning.

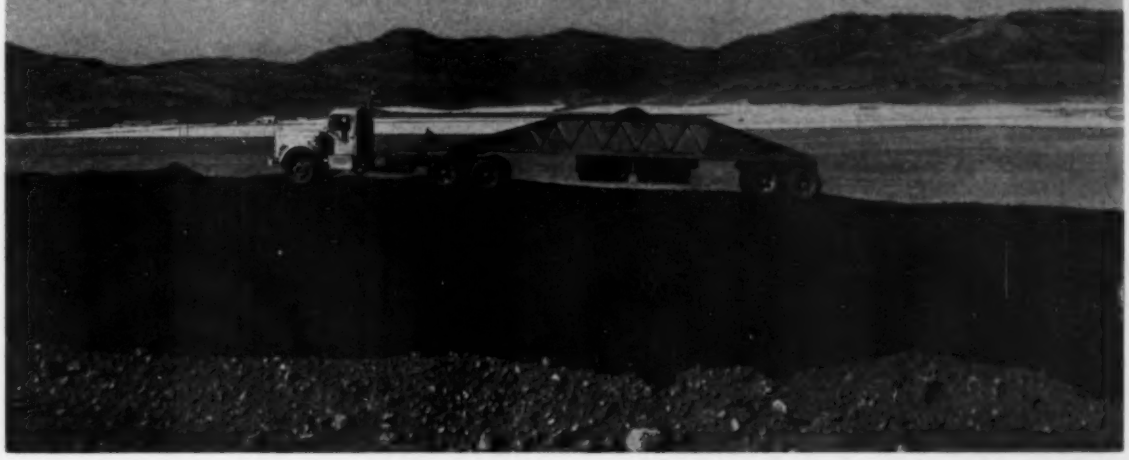
The primary advantages are:

1. Increased extraction, i.e., less ore in pillars.
2. Cheaper and faster stope preparation.
3. Dewatered stope muck.

The basic requirements are proper fragmentation, a scheduled maintenance program, and a satisfactory ventilation system. As production requirements increased, controlled ditching and the scheduling of mucking machines and trains became more important. The choice of a workable drawhole design, and proper follow-up on the cutout of the drawhole is of paramount importance.

Development of Monsanto's Western Phosphate Operation

by G. Donald Emigh



WESTERN phosphate resources which lie in Idaho, Montana, Utah, and Wyoming, have seen great expansion of activity in recent years. Growth has been two-pronged: expansion in fertilizer use, and since 1949, into the elemental phosphorous industry. Elemental phosphorous, produced in the electric furnace, goes largely into chemical making.

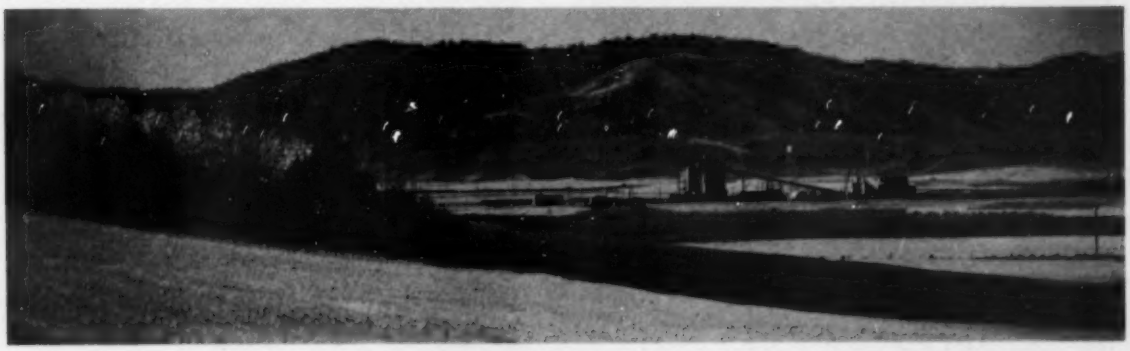
Phosphate ore was first recognized in the West about 1889. Production started in 1906, and for many years consisted of underground operations by small companies operating sporadically. In 1920 Anaconda Copper Mining Co. opened the underground Conda mine near Soda Springs in southeastern Idaho, and has maintained and increased its operations since then. About 1930 Montana Phosphate Products Co. began underground mining in Montana and has remained a steady producer.

In 1945 San Francisco Chemical Co., one of the oldest companies in the western field, initiated strip mining at the Waterloo mine near Montpelier, Idaho, and later at Leefe mine near Sage Junction, Wyo. In 1946 J. R. Simplot Co. began open pit mining at Gay mine near Fort Hall, Idaho. In 1950 Victor Chemical Works began underground development work in the area of Maiden Rock, south of Butte, Mont., and has since mined there. Monsanto Chemical Co. began open pit mining near Soda Springs, Idaho, in 1951.

G. D. EMIGH is Director of Mining for Monsanto Chemical Co., St. Louis.

Generally western phosphate occurs as layers interbedded in shales and limestones of Permian age. The phosphatic Permian formation, called the Phosphoria in Idaho, Montana, and Wyoming, and the Park City in Utah, is of economic importance over an area of 200 miles east and west, and 400 miles north and south. Phosphate beds extend beyond this area of northern Utah, eastern Idaho, western Wyoming, and western Montana, but beds are generally thinner, of lower grade, and of less importance.

Within the principal phosphate area there are marked changes in the Permian formation. Thickness of the formation is usually several hundred feet, with more limestone in Montana and Utah, and more shale and chert in Idaho. In any one area there are one or two phosphate beds of sufficient thickness and grade to be considered ore.



At top of page, furnace plant stockpile. Above, the electric furnace plant near Soda Springs, Idaho.

Northwest model 6 shovel with 1½-yd dipper loading at the Ballard property, about 16 miles north of the electric furnace plant near Soda Springs. Trailer trucks (also shown in picture at top of page 1077) were specially designed for the job. The 54-ft wheelbase on the 20-ton units is necessary to comply with Idaho load laws for highway transport.

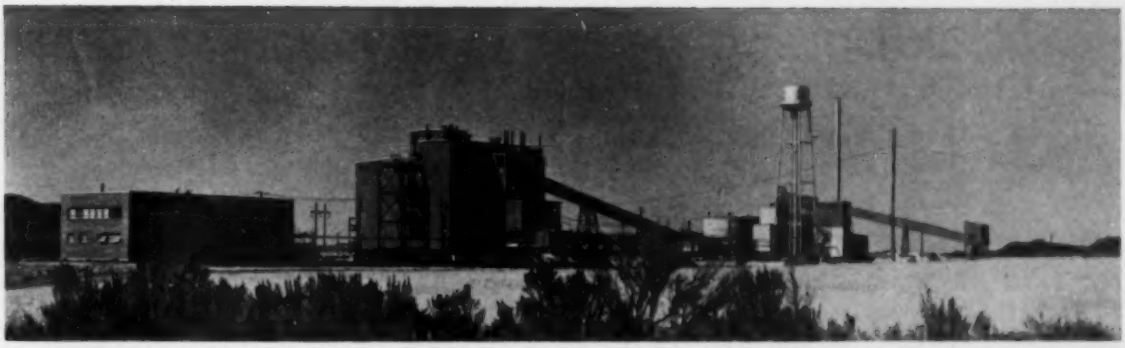
After the phosphatic Permian beds were deposited they were covered with many thousands of feet of sediments. Later, all formations were uplifted by mountain forming movements. The western half of the Phosphoria area was especially deformed by folding and faulting, and formations lie at all angles from vertical to flat. Mining practice obviously has been underground on the steeper dips, and surface on flatter dips where the ore beds are on or near the surface.

Prior to 1949 ore was mined for the fertilizer industry and required ore grades of 30 to 31 pct P_2O_5. This is about the highest grade mineable ore, but advent of the elemental phosphorus industry in 1949 made it possible to utilize lower grade ores.

After examining and sampling phosphate beds in the four states, Monsanto Chemical Co. decided in 1951 to build an electric furnace plant at Soda Springs, Idaho, near the southeastern corner of the state. Plant construction started in 1951 and stripping, loading, hauling, and stockpiling of ore at the plant site have been carried on during the summer months since 1951.

In the Soda Springs area the Phosphoria formation is about 400 ft thick. The upper half is largely a dark colored chert and is not phosphatic except for a few thin beds of no economic interest. The lower half of the Phosphoria is referred to as the Shale Section and is that part of the formation containing workable phosphate beds in this area. This lower half is composed of beds of gray limestone, and dark colored limey shales. Practically all of the beds are phosphatic; however, workable concentrations of phosphate occur in two places—near the top of the section and at the bottom. These are referred to as the Upper and Lower ore zones. A third zone, located near the top of the Lower zone, might be called the Middle zone, but for practical mining purposes it can be considered as part of the Lower zone. The Upper ore zone is about 10 ft thick, the Lower is 20 to 30 ft. The Lower zone contains the 5 to 6-ft high grade bed called *main bed*. This is the bed in

At Monsanto's Ballard property, shown here, the Upper ore zone has been largely eroded away, and operations are in the Lower zone. Stripping is with D-8 Cats and 17-yd Wooldridge scrapers. Special 20-ton trailer trucks haul to the plant.



Monsanto's electric furnace elemental phosphorus plant just north of Soda Springs. Shown left to right: office building, furnace building, water tower, steam plant (with stack), and at extreme right the furnace burden preparation area.

the Idaho area that has produced nearly all of the high grade ore required by the fertilizer industry. Above it are 15 to 25 ft of lower grade phosphatic shales. In the Lower zone there are several thin gray limestone beds. Two of these are quite persistent over southeastern Idaho, and are referred to as Cap Rock and False Cap Rock. Cap Rock limestone is about 2 ft thick and is between the main bed and the overlying lower grade shales. The False Cap Rock is 2 to 4 ft thick and is 15 to 20 ft above the Cap Rock limestone.

At Monsanto's Ballard property the Upper ore zone has been largely removed by erosion, and ore is obtained from the Lower zone. The formations on the property are relatively flat with dips being in the range of 5 to 15°. In laying out mining operations it was decided to strip the overlying waste beds with Cats and carryalls, and load the exposed ore into trucks with a shovel. It was also decided to move the ore, as much as possible, to the shovel with Cats in order to save on shovel moves and truck-road moves. The contracting firm, Morrison-Knudsen Co., does the stripping, loading and hauling of ore to the plant site. Monsanto exercises close supervision of the stripping and mining operations. It is interesting to note that there has not been a lost time accident since the start of operations.

The relatively small amount of diamond drilling done during the prospecting period on Ballard property had indicated considerable faulting, not detectable from surface showings. Before starting stripping in 1951 it was necessary to obtain more information on the faulting, and on the ore grades, in order to lay out a mining program. It was decided to try out a rotary drill and remove cuttings with air, so a truck-mounted Winter-Weiss drill was obtained and started operations in May 1951. Both 4½ and 5½-in. holes were drilled. This method of drilling the Phosphoria formation had not been tried before, but it proved to work beautifully. Within a few weeks the property had been drilled on a 100 to 200-ft spacing and enough information obtained to start stripping. Depths of holes were up to 200 ft.

Mining operations are carried on for about four summer months. The reason for a short season is two-fold: first, winter conditions make year-round operations impractical; second, large volume operations are cheaper. A contractor is used in order to eliminate company personnel and company-owned equipment standing idle for eight months. Between 350,000 and 400,000 tons of ore will be mined during the 1945 season.

Three layers of ore are mined and stockpiled separately. The separating 2 to 4-ft limestone beds (Cap Rock and False Cap Rock usually) are eliminated as much as possible at the mine with dozers. In places these limestone beds are fresh and hard, but at other places weathering has reduced them to clay. The surface ore, for the most part, is softened by weathering and by the time it is placed in the plant stockpiles has been largely reduced to —4 mesh.

Stripping is done with D-8 Cats and 17-yd Wooldridge scrapers. Rippers are used occasionally where hard limestone layers are encountered. Loading is with a Northwest model 6 shovel powered with a Murphy diesel ME-20 engine, swinging a 1½-yd bucket. D-8 dozers are used to move the ore to the shovel. Haulage for the 16 miles, mine to plant, is in Utility bottom dump trailers with Mack LH tractors equipped with Buda diesels. Hauling units have 54-ft wheelbase, necessitated by Idaho load laws for highway hauling of the 20-ton loads, and may have been the first heavy units brought into Idaho to conform with the recently enacted load laws.

Mining operations are two-shift, and stripping operations one-shift. Two-shift operation means poor light conditions at the beginning and end of the day and at such times there is some difficulty in selective mining because appearance of the ore is used to a great extent as a guide.

At the plant the three grades of ore are stockpiled individually by the trucks pulling on top of the piles, and bottom-dumping their loads either on top of the pile or while going off one end. A dozer on top of the pile spreads the ore.

Monsanto's electric furnace plant is 2 miles north of Soda Springs, Idaho. Construction started in July 1951 and it went into operation in the fall of 1952. Plants of this type reduce the elemental phosphorus by smelting the ore with coke in a closed electric furnace. The coke reduces the phosphate in the ore which volatilizes with the furnace gases. Gases are first treated in electrostatic separators in order to remove entrained dust, and then are cooled with water sprays, which lower the temperature sufficiently for the phosphorus to condense and come out as a liquid. Phosphorus is extremely active when exposed to oxygen so as soon as it is produced, either as a gas, or later as a liquid, it is kept away from air. After it has been brought down as a liquid it is always kept under water whether in storage tanks or in transit in tank cars.

Elemental phosphorus is shipped to Monsanto plants in the Midwest to be burned to phosphoric acid and then reacted, usually with soda ash, to produce a sodium phosphate which is further processed into various phosphate chemicals.

Rock Bolting Finds Wide Application

"The application of rock bolting as a means of ground support is limited only by the resourcefulness and ingenuity of the mining engineer."

by Edward Thomas

ROCK bolting, third great change in postwar underground mining, is inseparably tied to the other two changes: better drilling and mechanized mining. Mechanized mining provided the impetus, when conventional supports restricted maneuverability of equipment—while carbide-tipped bits lent the means of drilling required holes cheaply. With the need and the tools at hand development snowballed. In just six years, from January 1948 to January 1954, some 800 mines adopted bolting for systematic support, and over 100 of these mines now use rock bolts exclusively. Simultaneously, foreign mines have adopted the idea, but to a lesser extent.

What held back this major change? The concept of fastening loose material to unbroken and apparently solid material is probably as old as mining itself, and sporadic use has been reported from virtually every mining region in the world. But falls of rock have always been the greatest source of both fatal and nonfatal accidents in underground mining and miners have understandable reluctance to work under roof or back material if they even suspect that it might be hanging like a "sword of Damocles" —supported only by a thin bolt.

Another factor was that until the advent of carbide alloy insert bits, the cost of drilling holes in hard rock deterred serious consideration of bolting as part of systematic ground support. The tendency was to limit drilling wherever possible.

And last, development of mechanized mining methods and equipment only recently progressed to the point where a new systematic means of ground support became a necessity for efficient operation. In fact, mechanization was being retarded by the use of bulky conventional supports that restricted the maneuverability of equipment.

Principles Involved

The basic difference between bolting and conventional support is sometimes overlooked. Successful bolting depends upon being able to make the ground itself an integral part of the support structure, whereas with conventional supports one assumes that failure of the ground is inevitable and prepares to support it within certain limits.

To reinforce the ground, over 95 pct of the bolts now in use consist simply of steel rods either ¾ or 1 in. diam and 2 to 8 ft long, having an anchoring device at one end and a nut and a bearing plate on the other. The diameter of the drill hole is large enough to accommodate the fastening device, and by a bearing plate, placed over the collar of the hole, the rod is loaded in tension. Inasmuch as the rod between the anchoring device and the bearing plate does not usually touch the perimeter of the drill hole, it is best to assume that no force other than tension is applied between these two points. This tension in the rod compresses the rock between the anchor and the collar of the hole. The mechanics of the transferal of this compression to effect reinforcement of the rock against other forces present is a research problem for stress analysts; however, the method has been proven by experience to be effective. To the practical engineer, however, this conventional type bolt seems a most inefficient way of reinforcing ground such as the hanging or steeply pitching walls of stopes or shafts. Here the dominant force may have a large shear component, and for this a usable bolt is needed that will bear against the entire length of the perimeter of the drill hole.

In the light of accumulated experience these classifications of bolting for engineering utilization appear to be logical:

1. **Suspension**—pinning a loose piece of rock to ground considered self-supporting.
2. **Beam building**—using bolts to cause a series of beds to act as a monolith so that stresses are reduced to a value that the beds can safely resist. This type of bolting is used principally in rectangular openings.
3. **Reinforcement of the skin areas** of an arched opening to provide additional support to counteract the effects of tensile, compressive, and shear stresses.
4. **Reinforcing walls** against stresses that are totally or dominantly shear and compression.

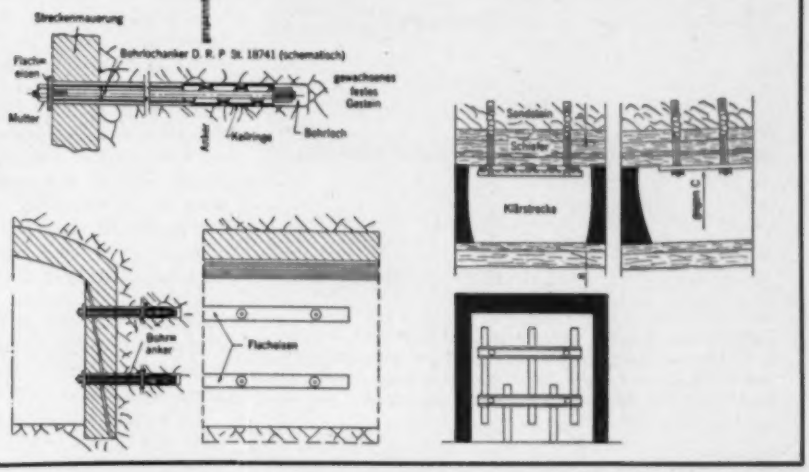
Suspension Bolting

This type of bolting was the prototype of modern rock bolting. At the St. Joseph Lead Co. mines, which deserve a major portion of the credit for

Slotted type rock bolts are used here with 4-in. channels at TCI Div. of U.S. Steel Corp. Angling rib bolts and tying them with channels affords advantage over vertical bolts with individual bearing plates in difficult areas. Reference 5.

EDWARD THOMAS is Chief, Roof Control Section, U. S. Bureau of Mines, Washington, D. C.

These German drawings are from the first known published account of bolting in coal mines, *Versuche und Verbesserungen geim Bergwerksbetriebe in Preussen Wahrend des Jahres 1918*. This short account of experiments prior to World War I, in Upper Silicica, now Poland, discussed support of concrete reinforced roadways in longwall areas, and bolting of weak shale to self-supporting sandstone above. The tests were brought to a halt by the war and were not resumed. Drawing from J. Schlickau, consulting engineer, Belfry, Ky.



pioneering in bolting, this type of bolt is called a *shin plaster* and is used with no particular pattern to secure single pieces of loose or questionable material in areas that are otherwise considered self-supporting. Such bolts should be strong and have enough anchorage to support the estimated weight of the ground under question with a safety factor of at least 3 to compensate for uncertainties of anchorage and load.

Beam Building

Roof bolts may be used in stratified and loosely consolidated rocks to bind them together to act as a single beam rather than a series of separate beams. However, with conventional slotted type and expansion shell bolts an undesirable disturbance of the strata is caused by bolt action at the anchorage.

Properly anchoring these bolts to develop the full strength requires tremendous thrust against the rock sides of the drill hole. The thrust normal to the long axis of the hole tends to create a fissure in the rock at the anchorage; this fissure extends parallel to the stratification, and will usually be parallel also to the roof surface of the passageway supported. In pattern bolting these fissures from individual bolts connect to form a continuously weak horizon that segregates the monolith, formed by bolting, from the rock above.

Therefore, with beam building bolts it is important that no dependence be placed upon suspension of a weak shale by drilling a few inches into a massive and strong formation above. The bolts should be long enough so that total thickness of the resulting monolith renders the beam self-sufficient.

It is believed that over 80 pct of current bolting being done in horizontally bedded mineral deposits is in the category of beam building.[5, 8, 9, 10] There have been two interesting foreign developments along this line.

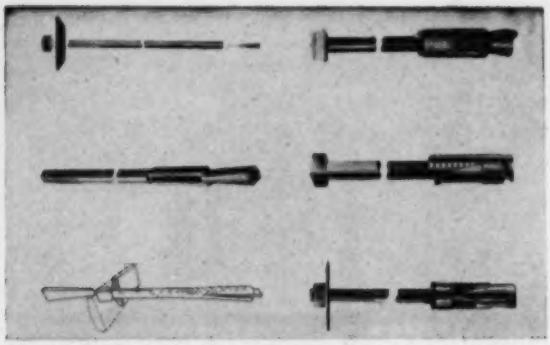
An ingenious method of roof support is practiced at A/S Sulitjelma Gruber in Norway.[10] The ore is 1 to 2 meters thick and where it dips 10° to 30° is mined by scrapers working on a long face which closely resembles longwall coal mining. Conventional rock bolts with cribbed pillars of schist blocks are ordinarily used, but where extra roof support is required ½-in. wire rope is *sewed* and grouted into vertical drilled holes. Several interesting and promising experiments have been made in the coal mines of Scotland and Nova Scotia in compacting roadway roof and floor in advancing longwall workings.[11]

Reinforcement of the Skin Area of an Arched Opening

General mining experience has shown that a mine opening having an arched roof or back is less likely

Pattern bolting at St. Joseph Lead Co. uses these 8-ft slotted type angled bolts with 4-in. channels as bearing plates against thinly bedded limestone back. Ref. 1.

Pattern bolting at the TCI Div. of U. S. Steel Corp. called for 6-ft slotted type bolts on 6-ft centers, laterally and longitudinally, to support a thinly bedded shale roof. Ref. 4.



Typical expansion shell and slotted steel bolts used in the U. S. Also shown is the wooden bolt developed by Lanier, and used in the Coeur d'Alene. Upper row shows three expansion shell types, lower row two of the slotted steel type.

to require artificial support than a rectangular opening in the same material. Moreover, this indicates that the circular or semicircular opening is the most desirable.

Necessity for this use of the roof bolt comes from two causes: the nature of the stresses about mine openings (in this connection it should be kept in mind that rocks are very weak and unreliable in tension) and the irregular shape of an opening made by conventional tunnel driving methods and the fracturing and shattering in the sides of an opening caused by blasting incidental to tunnel driving.

When any mine opening is made, the most intense stress is encountered within a short distance of or at the skin of the opening. The effects of the opening on rock stress, however, decrease very rapidly with the distance into the ribs or back of the opening. It is safe to assume that at a distance into the back or ribs equal to the long cross-sectional dimension of the opening the increased concentration of stress caused by the opening is negligible.

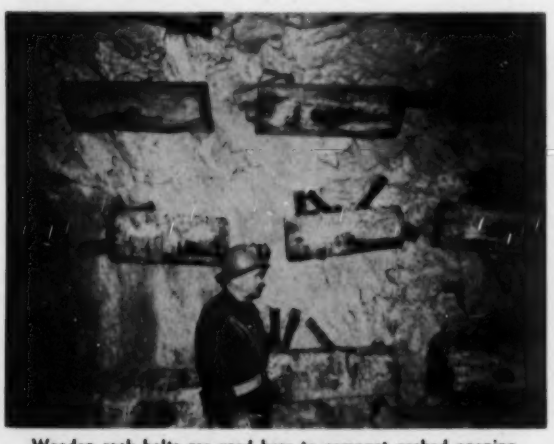
Whether or not the skin stress will be tensile or compressive is determined by the degree of arching, the ratio of the width of the opening to its height, and the ratio of the vertical load to the horizontal load on the opening, as well as orientation of the principal axes of the opening to the direction of the acting loads. Of these factors, the magnitude of the horizontal load on the opening is almost always unknown. Consequently, one can never be sure that the back or roof of the arched opening will not be in tension. Even in the cases where the roof is in tension, however, it is certain that a relatively short distance in from the surface the rock is in compression or at worst under very low tension. Consequently, relatively short rods, rarely longer than 6 or 8 ft, will reach rock from which it is comparatively safe to suspend any loose ground at the surface of the opening.

Tensile stresses are not the only cause of ground failure in arched openings. Compressive stresses may cause flaking and sloughing of the ground from the skin. But again, the intensity of these stresses decreases rapidly with distance into the rock, and systematic bolting may be used to suspend major pieces of rock that break from the skin due to compressive stress.

In tunneling, if an opening is being made in rock that has enough inherent strength to withstand the load of the superimposed strata, and if it were possible to cut exactly to a semicircular line (see "A" in figure on page 1083), no artificial support would be required. However where explosives are used this is not possible. Blasting produces a jagged break line, exposing and opening bedding or jointing planes and produces fractures that extend into the back and ribs.

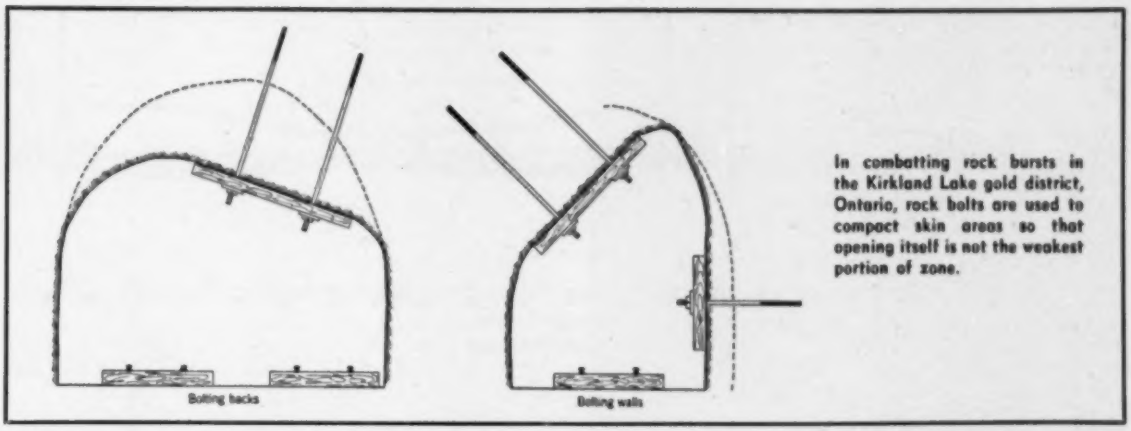
In mining, the actual arch line is outside the maximum overbreak or "B" line, and the rock between the "A" and "B" lines may require artificial support. The radius of this "B" line depends upon the effect of the explosives used, but a slotted type bolt whose penetration is the difference in radius between the "A" line and the deepest penetration of the overbreak multiplied by 2 appears to offer a satisfactory safety factor.[12] This is borne out by the published information on large bore tunnel driving, and in the drift and crosscut experience of Anaconda Copper Co. at Butte.

When sloughing or spalling occurs about the skin surface of an arched opening owing to excessive skin stress or to weathering, bolting has definite value in compacting and reinforcing this skin area. Although the conventional slotted-type bolt was used successfully for this purpose in the Bureau of Reclamation's Duchesne tunnel in Colorado, it is believed that, inasmuch as ordinarily the only possibility is to place bolts normal or nearly normal to the skin surface,



Wooden rock bolts are used here to compact arched opening in loosely consolidated and shaly quartzite. It is not feasible to anchor steel bolts in this rock with devices now in use.

Typical bolting in Delaware Aqueduct Tunnels used No. 5 mine ties and angle washers for roof support. This operation pioneered bolt use in aqueduct work. Ref. 12.



In combatting rock bursts in the Kirkland Lake gold district, Ontario, rock bolts are used to compact skin areas so that opening itself is not the weakest portion of zone.

a bolt that fills the entire drilled hole would be more efficient. The wooden bolt developed by Sterling Lanier, Jr., is used for this type of application in the Day mines[13] in the Coeur d'Alene district where it stabilizes the skin area of arched drift openings in a loosely consolidated and shaly quartzite. It would not be feasible to attempt to anchor a steel bolt in this rock using any fastening device now on the market; moreover, at least two thirds of the bolt length bears against the drill hole circumference.

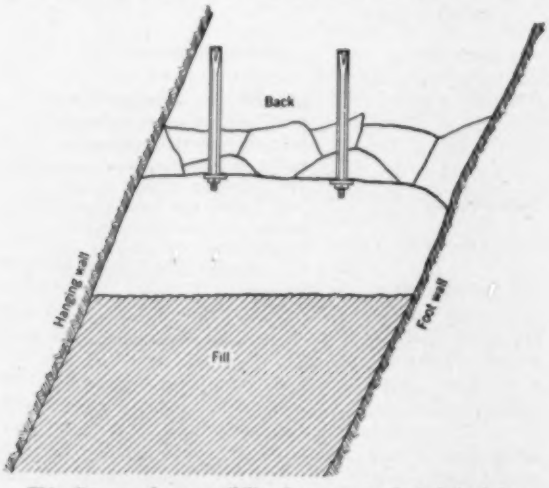
At Lake Shore[14, 15] and Macassa mines in the Kirkland Lake gold mining district of Ontario, rock bolts are utilized in combatting the hazard of rock bursts. This is being done to compact the skin areas of arched openings attempting to supply enough additional reinforcement so that the opening itself is not the weakest portion in a mass of ground being subjected to extraordinary loads. Also, where past experience indicates that rock burst is inevitable during the active life of a drift or crosscut, bolts are used to support a cushioning structure intended to contain the flying rock.

Reinforcing Walls Against Shear and Compression

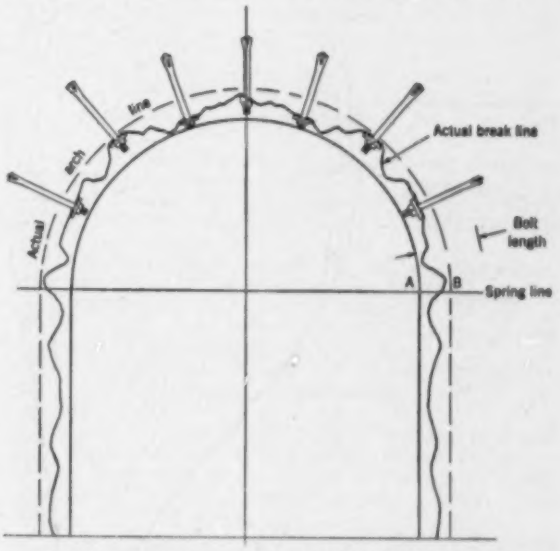
In shafts or steeply pitching stopes there exists the probability of resisting a load resulting from hundreds of feet of an unsupported vertical span, regardless of whether walls are of sedimentary rocks or the contact zone rock of an igneous formation. A bolt freely suspended between an anchoring device and a bearing plate and contributing only tension to the structure and in a direction normal to application of the load does not appear to be the most efficient device to use for compacting such a skin area. What is needed is a bolt that will better constrain motion in a vertical direction. It is quite true that a conventional bolt will resist shear stresses after the ground starts to move, but then the load can no longer be considered static and the ground has started to break. Failures of this type have been experienced in both the U. S. and Canada.

Nevertheless, where the unsupported span is not too high, the effect of strain resulting from the vertical component of this column has been combatted effectively with the ordinary slotted-type bolt. This action was described by V. D. O'Leary in his paper before the National Safety Council meeting in Chicago on bolting at the Anaconda mines at Butte.[16]

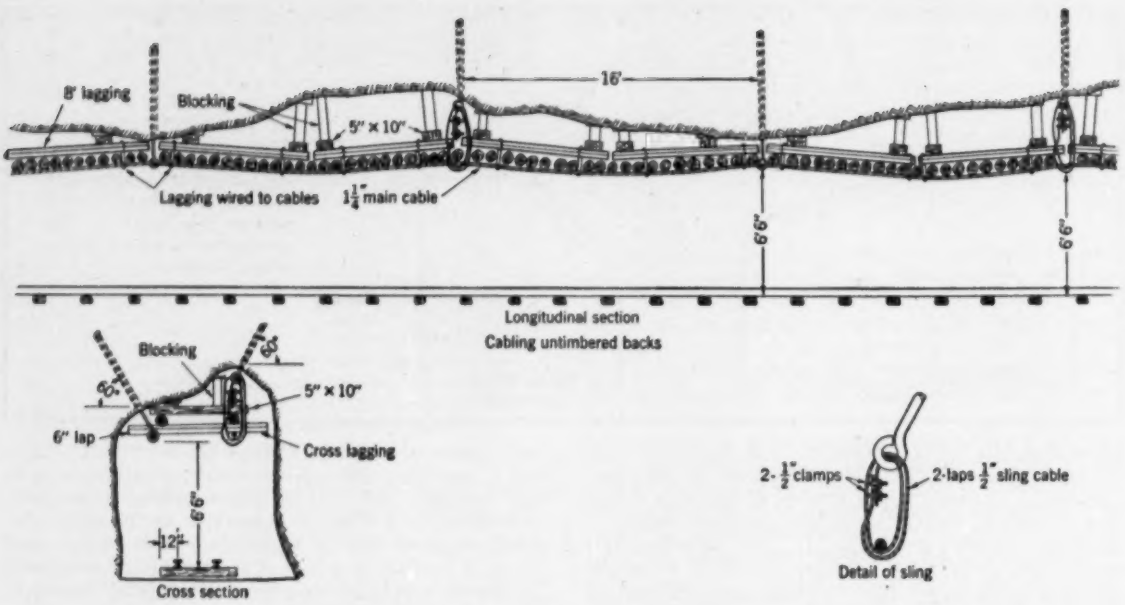
Virtually all the veins in Butte have the problem of a badly jointed hanging wall. This false hanging wall, as miners call it, is 1 to 6 ft thick and makes it almost impossible to mine without a timber support. Horizontal cut-and-fill stopes without timbers were tried, but because of an inadequate supply of



This diagram shows possibility for support of weak back in cut-and-fill stoping. Bolts support back during mucking cycle and are blasted out in the next cut.



In bolting applied to tunneling, line "A" marks ideal opening, never approached when explosives are used. Actual arch is outside maximum overbreak or "B" line, and rock between "A" and "B" requires support.



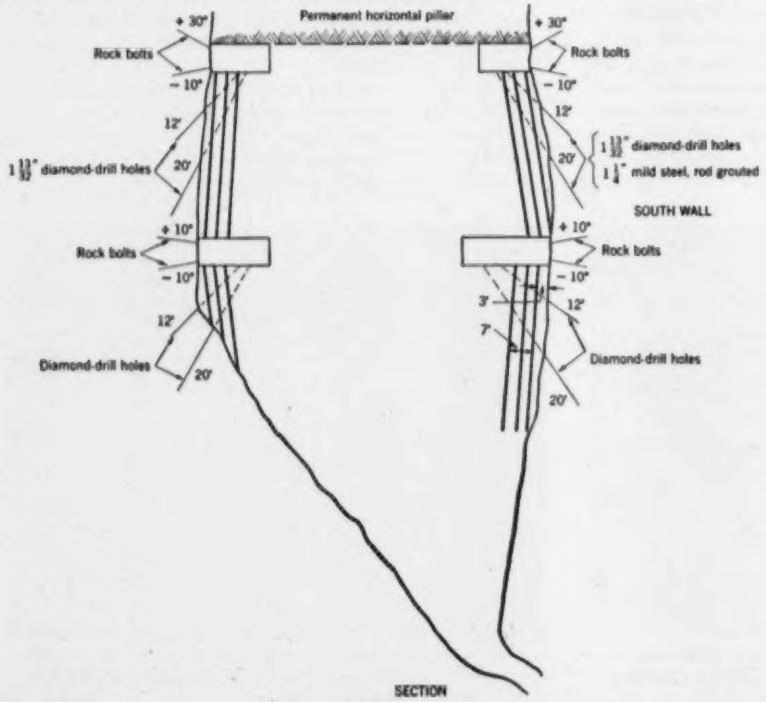
In Kirkland Lake district, Ontario, rock bolts are used to combat rock bursts in two ways: to compact skin area, and to support a cushioning structure containing flying rock. Drawing shows details of structure to protect personnel.

waste to fill the mine workings immediately, a great deal of trouble was caused by waste sloughing from the walls to dilute the ore and in some instances loss of an ore block due to caving.

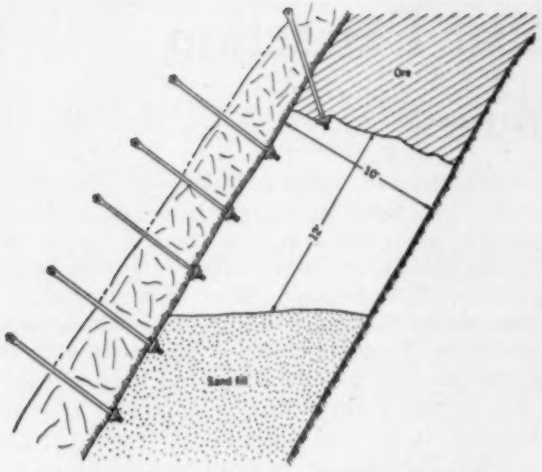
Study of the hanging wall seemed to indicate that the heavy blocks of ground moved along the joints in a sliding, downward fashion. This movement set up a shearing stress along the contact between the ore and the hanging wall rock so that the ore would be dragged down and eventually fall. If the sliding movement in the hanging wall could be temporarily stopped until the fill was placed against it, it seemed reasonable to suppose that stopes could be operated without timber. Rock bolts proved to be an answer.

The slotted-type bolt is being used successfully in several other ore mining districts to compact the walls of cut-and-fill stopes where the vertical distance between the back and the fill is not excessive.

In 1951 and 1952 an 11-ft sq airshaft 219 ft deep was sunk through sedimentary rocks at the Stansbury mine of the Union Pacific Coal Co., Rock Springs, Wyo., using 1-in. diam, 6-ft long slotted-



At Quemont Mining Corp. nearly vertical waste walls are being experimentally compacted before actual mining of ore. This operation is part of sublevel system, and bolts are cemented in place before mining begins. From information by R. J. Dionne, chief engineer of Quemont.



At Butte rock bolts have been used in cut-and-fill stopes to tie ore to rock in hanging wall while fill is placed. Bolts reduced or eliminated the shearing stress along ore-waste contact. Ref. 15.

This 11-ft airshaft at Stansbury mine, Union Pacific Coal Co., Rock Springs, Wyo., used 6-ft rock bolts installed over mesh wire to hold shale and sandstone walls. See text for cost data.

type bolts on 34-in. centers horizontally and 27-in. centers vertically to compact shale and sandstone walls. The bolts were installed over 12½-gage V-mesh wire 58 in. wide, with an overlap of 2 in. The bolts and wire mesh were installed as the sinking proceeded, and afterward it was decided to gunite the shaft to a minimum thickness of 1 in. outside the wire mesh.

The cost of bolting and installing wire mesh, including labor and material, was $35.52 per linear ft of shaft as compared with an estimated cost of $54.66 if wooden timbering were used in the shaft. The total cost, including guniting, was $74.00 per foot of shaft compared with an estimated cost of $68.41 per foot for concrete lining. However, the estimated cost of the concrete lining did not include the indeterminate cost of temporary support necessary preparatory to pouring the concrete.

At the Quemont Mining Corp. mine, in connection with a sublevel mining system, an interesting experiment is being tried wherein the nearly vertical waste walls are compacted before actual mining of the ore. The walls of the sublevel stopes were sloughing as the area of exposed wall increased. Inasmuch as the development comprises the major expense in this type of stoping, it is desirable to obtain maximum ore recovery with minimum dilution.

The program is to drill extra holes along the wall and cut down the powder concentration in all wall holes to minimize shattering. Second, tubular rock bolts are cemented into the walls on 6-ft centers midway between the blast-hole rings. At the sublevel elevation two 12-ft slotted-type bolts are placed in each row, with 3½-in. diam washers countersunk into the rock.[17]

Between sublevels 1 13/32-in. down holes are drilled with a diamond drill, using a noncoring-type bit. In these holes 1¼-in. diam mild-steel rods (in the future, 1-in. standard pipe will be used) 12 and 20 ft long are grouted into the hanging wall. The 12-ft rods are single pieces, and 20-ft rods are two 10-ft pieces with a tapered flush coupling.

Immediately after the diamond drill hole is completed a calculated amount of a stiff cement slurry is poured and carefully placed by slowly withdrawing the string of rods from the bottom. The rock bolt is then dropped into the hole and pushed into position through the slurry. A quick-setting aluminous cement is used. Preliminary results look promising.

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Longhole Drilling for Pillar Extraction At the Torbrit Silver Mines

Many persons hold the opinion that the metal industries are fortuitous and that the occupation is one of sordid toil, and altogether a kind of business requiring not so much skill as labor. But as Agricola says, *It appears to be far otherwise.* This still holds true today and the miner is forever seeking means to combat rising costs, to find an easier method to break rock, to decrease costs, and to increase safety. The study of longhole drilling and blasting of pillars offers some relief in meeting these problems. Investigation into the use of tungsten carbide bits, coupled alloy steel rods, and the heavy Leyner percussion drill led the Torbrit mine to adopt this type of drilling for pillar and sill recovery. It has been further developed for regular stoping.

by Harry Bapty

LONGHOLE drilling with a heavy type percussion Leyner drill, jointed steel rods, and a tungsten carbide insert bit was investigated while drilling and working a quartz barite brecciated rock. Procedure and practice were developed and its suitability for tackling pillar and sill recovery proven and extended to regular stoping at Torbrit.

Torbrit, as the name implies, is a contraction of the Toric property and the Britannia Mining & Smelting Co. Ltd., the former owners. This silver property, with minor amounts of lead, zinc, and rarer sulphides of little or no economic value, was discovered in 1916 from an outcrop in the Kitsault River, 18 miles north of Alice Arm Post Office, in the Portland Canal Mining District, B. C.

Geology

Ore in the Torbrit mine occurs in an irregularly folded vein about 120 ft wide. Striking roughly northwest, dipping about 40°, and plunging down toward the northwest. Only the richer parts of the vein are mined, sometimes to a width of 80 ft.

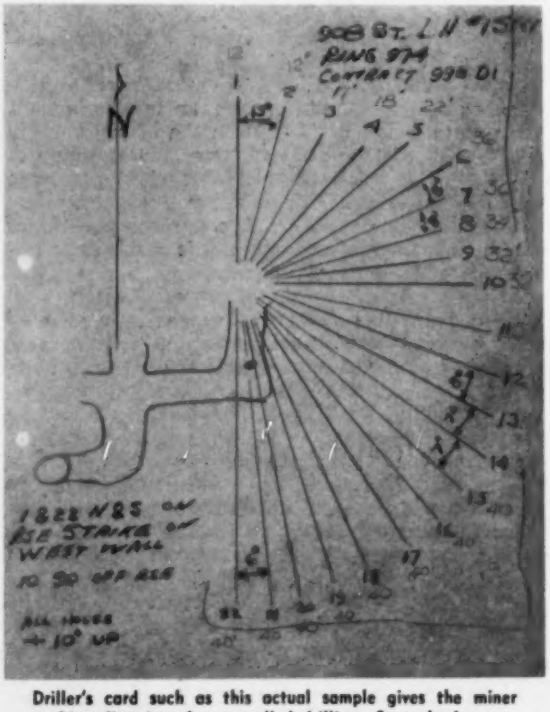
Native silver, argentite, ruby silver, and other silver bearing minerals are contained as a hypogene replacement in quartz barite brecciated vein. The deposit was formed in the Hazelton Jurassic series, which have been folded and altered.

Pillars

Today pillar extraction is recognized as an end phase of clean mining methods and is universally practiced in all operations where the recovery cost is economical and not prohibitive. The Torbrit system of mining varies from straight shrinkage stoping to open stoping. Stopes have a maximum length of 50 ft with a 25-ft pillar interval. This leaves a pillar block 25 ft thick and 50 to 100 ft long between levels. Raises are generally driven in the center of the pillars and subdrifts broken into the stopes. These raises serve as entries to the stopes on either side of the pillars.

This system of mining leaves a large tonnage of ore in sills and pillars. Ore is recovered by retreating from the east or top of the plunging orebodies towards the west, and pillar recovery is kept just ahead of the underlying sill recovery. Sectional steel with 2-in. tungsten carbide bits is used at present to drill horizontal and up holes.

Pillars between sills are recovered by first undercutting the bottom of the pillar above the sill, then drilling horizontal rings of holes up to 45 ft long from hanging to foot wall from inside the pillar raise. The holes are laid out to give a 6-ft burden on the end, and 5, 4, or 3 ft between rings. Since the 2N-2S pillar has been blasted and secondary blasting figures are available, the burden between rings has been decreased to 4 ft and in doubtful places to 3 ft. In the first longhole drilled pillar, several rings were kept ahead of the blasting. These holes were loaded with 1⅛-in. 40 pct Forcite gelatin powder and detonated with Primacord. Latest practice is to load the end of the hole with 1¼-



Driller's card such as this actual sample gives the miner working directions for controlled drilling. Spot checks compare actual hole depths and direction with theoretical layout.

HARRY BAPTY is Mining Engineer, Torbrit Silver Mines Ltd., Alice Arm, B. C.

Plan compares random vs controlled drilling. Holes shown left of centerline were controlled with some random. Holes at right were random. Results of tests determined that control was vital to adequate breaking.

in. 40 pct Forcite and detonate with millisecond delays.

Mapping

Pillar maps are prepared from surveyed stope maps. Plan outlines are contoured at 10-ft intervals showing each lift for the ring drilling. For clarity the same pillar body is vertically sectionalized in 10-ft north-south section maps to help supervise drilling. From these work maps, a diagram is made up on a piece of cardboard about 8x11 in. On this is drawn the raise opening, and the drill hole layout. This layout is a guide to assist the miner in drilling. This card shows direction of holes in the raise and each hole length is clearly marked. The miner reports his ring number by elevation, hole number, and the length drilled for each hole on his daily time card at the end of each shift.

Drilling of Pillars

Ring collar elevations are marked underground for the miner and are tied into known survey plugs. The mine superintendent or a mine engineer hands the miner the new cardboard ring layout showing the drilling plan for the new ring. The card is oriented horizontally at the ring elevation and the miner shown his No. 1 hole and the approximate collar positions of the remaining holes. Normally the No. 1 hole is a north-south hole corresponding to the strike of the raise. To foster the drillers' responsibility and interest in their work, the miners are encouraged to drill the odd extra hole if they think the burden is excessive at the hole extremity. The miner is also instructed where he will hit the greenstone hanging or footwall and warned to stop his drilling at these points. Similarly, he is shown what holes may break into old workings. When he senses he is getting close to a break-through and the planned length of hole has not been reached, the miner is instructed to terminate the hole.

In experimental work it was found flat holes are most likely to mud and all holes wandered after drilling about 15 ft. In order to keep the drilling gear as simple as possible in keeping with existing drilling methods, it was decided not to adopt the water swivel for flat or down holes. Consequently, all so-called horizontal ring drilling is actually drilled up at 10°. Hole dips, ring elevations, and hole lengths are periodically spot checked by the engineering staff and appreciable discrepancies from the theoretical layouts are reported to the mine superintendent.

When drilling toward an area removed by a longhole blast, and located where no close inspection is possible, all longholes are terminated 6 ft short of the previous blast to avoid drilling into possible missed holes.

Drilling Equipment

For longhole drilling the 2-in. Carset Jackbit manufactured by Canadian Ingersoll-Rand is used. This type of bit offers great resistance to wear, and is used until 3/32 in. has been dulled off the striking face, then resharpened.

Hollow drill steel used by the Torbrit mine is a product of the Atlas Steel Ltd. of Welland, Ont. The 1-in. quarter-octagon steel originally used has been replaced by 1⅛-in. round Atlas Chippewa hollow shanked steel.

This steel is softened for forging and machining by heating in a Denver oil-fired furnace to 1850° to 1900°F and squared up in a Denver sharpener. The steel is annealed in powdered lime.

Machining is accomplished by a mounted turret in the lathe capable of cutting a type 2 thread for the Carset bit and type 1 thread for the couplings.

The following lengths are made up and a reserve of each length kept both in the mine and at the shop:

Leader (adapter) cut from 1¼-in. drill steel, 8 to 12 in. long, with type 1 thread on one end and type 2 thread on other end
2-ft shank to coupling
2-ft coupling to coupling
4-ft shank to coupling
4-ft coupling to coupling
6-ft shank to coupling
6-ft coupling to coupling

Originally only new steel was used for the above lengths and the broken returns made up into conventional lengths for regular mine steel. The practice now is to reuse and rethread broken ends and send the steel underground several times.

Some of the original work on coupled steel for longhole drilling was performed at Wingdam, B. C., in 1932. At this property the Australian deep lead system was practiced and the longholes were necessary for water drainage.

At Torbrit, the mine coupling replacement and steel breakage at the coupling joint has been high. Filleting the steel at the end of the thread is now practiced as suggested by P. W. Billwiller in his paper, *Introduction of Tungsten Carbide Bits at Britannia Mines.*

Summary of Tests

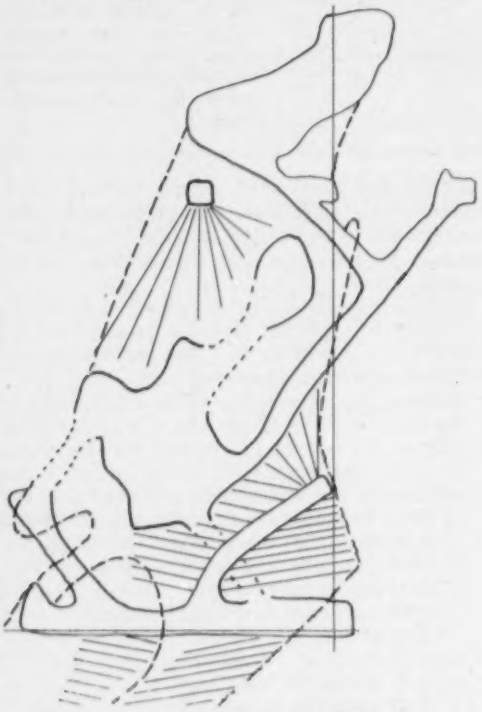
Longhole drilling at the Torbrit mine was commenced in December 1949. The following equipment was purchased for this experimental work from the Canadian Ingersoll-Rand Co.:

Bits	2-in. Carset bits with type 2 thread
Steel	1-in. quarter-octagon alloy steel in the following lengths: 2 ft 8 in., 4 ft 6 in., and 6 ft 6 in. with a type 1 thread on one end and shanked on the other. The 6-ft steel had a type 1 thread on each end.
Couplings	Type 1 thread

The above equipment was used in an Ingersoll-Rand DA-35 drill.

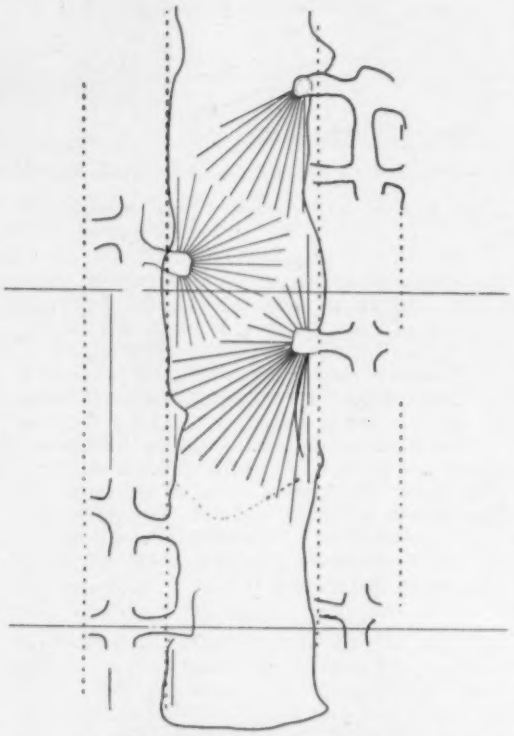
A series of six holes ranging in length from 22 to 54 ft was drilled in an 80° arc starting from the pillar wall. All holes were drilled from the same setup and were drilled at from +3° to 10°. Five of the six holes were blasted. The sixth hole was considered to have too little burden and it was decided that it was unnecessary to load it. One

Torbrit Longhole Drilling Patterns



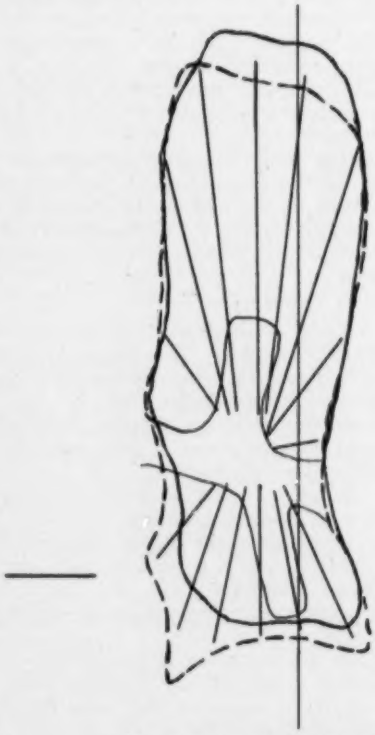
Percussion and Diamond Drilling

Upper holes were diamond drilled, lower ones in sill were percussion drilled.

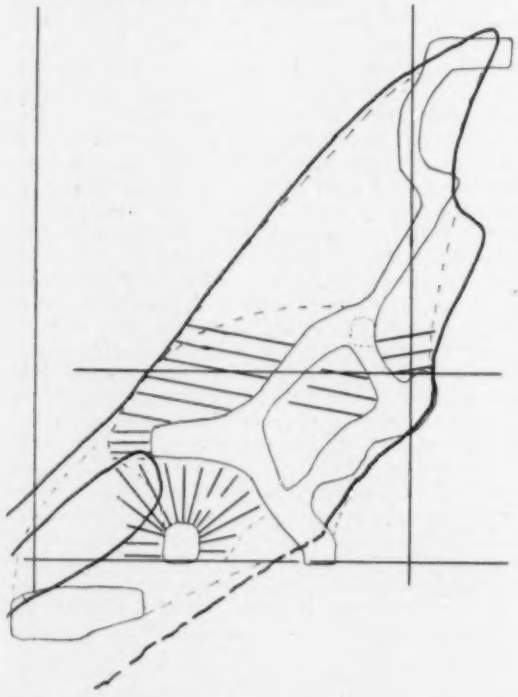


Stope and Pillar Longholes

Longhole drilling of the 908 stope from three stations. Plan shows two pillars, between dotted lines.



Plan of Holes for 2N-2S Pillar



Section 8200E

Plan and section show original longhole drilling layout used for the 2N-2S Pillar.

hundred and fifty lb of 40 pct Polar Forcite dynamite was used and blasting was done electrically using No. OEB caps, breaking approximately 350 tons of ore.

Five steels were broken in the test, all at the thread inside the coupling. Three bits were used and all were apparently capable of considerably more footage. All holes tended to turn down and holes started at +10° were approximately horizontal at the end of the 54-ft run.

Torbrit Pillar and Sill Drilling

From the results obtained in a long series of experiments, it was decided to lay out the 2N-2S pillar for longhole drilling with provision to change this type of drilling program as more experience was gained. First, it was necessary to sill out under the pillar to give a breaking space and longhole drilling commenced on the detailed drill plan. The original plan called for three rings to be completed and then the lowest ring blasted. These rings were drilled with 5-ft vertical spacing as a trial. The first ring was loaded and blasted with electric caps on Sept. 23, 1950, and during the month 25 pct of the pillar was blasted. This first ring blast produced favorable results so it was deemed advisable to proceed with the drilling and carry out the drilling program well ahead of the blasting. In the next blast several rings were blasted simultaneously with electric caps and no holes lost above the loaded portion. All results so far exceeded expectations and encouraged the management to try to drill off the whole remaining pillar and blast when drilling completed. Drilling was completed by the end of October 1950 and the remaining pillar blasted near the end of December. In this pillar, 12,439 tons of ore were broken for 6148 ft of drilling and 4276 lb of powder giving a powder factor of 0.343 lb per ton.

Considerable oversize rock was contained in the muck which required plugholing and sandblasting. Primary and secondary blasting of this rock amounted to more than the regular stope blasting factor of 0.8 so it was decided to increase the footage drilled per ton from 0.494 ft per ton to 1.0 ft per ton, or near the stope figure of 1.5 ft per ton. This was accomplished by spacing the rings closer together and lessening the hole end burden to 5 ft. Thus the ring spacing was reduced from 5 to 4 ft and in some doubtful spots the rings were further decreased to 3-ft spacing.

Immediately after the 2N-2S pillar was successfully blasted, the sill above the 1300 level and over the No. 2 South stope and the 11-1 stope was tackled as a longhole project. The ground here was not particularly thick as it was well opened up by box holes. Some random drilling was tried to discover what type of a pattern would result. Later a survey was made of this particular area and it was found that in some places the ground was inadequately covered and in places additional drilling was necessary. As a result of this experience longholes are now laid out according to plan and all drilling controlled as far as possible.

The possibilities of this type of drilling were evident as this work proceeded and it was decided to drill off all pillars when stoping was completed on the pillar boundaries. At the end of 1951, there were 25,377 tons drilled off awaiting a suitable time for blasting and 27,063 tons have been blasted for 19,556 ft of drilling.

To insure complete blasting to the toe of the holes, double priming was attempted. This was accomplished by running in a string of Primacord into each hole with the first batch of powder and tamped in to the end. A No. OEB Cap was inserted at the collar of each hole. Apparently no misses have occurred and it is believed from examination of the muck being drawn that all holes have been blasted throughout their entirety. To carry detonation one step further, millisecond ignition of powder is now in use. As more pillars are removed and ground support becomes less the desirability of less ground shock obtained with millisecond blasting is to be commended.

The No. 11-1, 2S, pillar of 8705 tons was drilled down by noncoring diamond drilling. Drilling per shift by drill runners ran from 77 ft per shift to 13 ft as against 288 ft per shift down to 98 ft per shift for longhole drilling by percussion drill methods while other costs remained about equal. These results have discouraged further diamond drill blast hole drilling for the present, except for down holes and those holes which cannot be reached by longhole steel.

Conclusions

It has been found that the ordinary mine equipment can be adapted for percussion drilling of long blast holes with little additional outlay for equipment, as the list of equipment on p. 1087 indicates.

Torbrit is a firm believer of rewarding good work by an attractive incentive bonus system. The bonus paid per foot drilled on longhole work has been sufficient to attract the better class driller. It has been found that he is anxious to follow instructions and adapt himself to this type of work. The working place is visited during the morning rounds at which time hole alignment and hole elevations are checked by the mine superintendent. The whole shift is spent on actual drilling and moving delays kept to a minimum.

Drilling the whole pillar in one continuous operation constitutes a multitude of setups. Fortunately, each setup is only a move of a few feet, thus, the maximum amount of time is devoted to drilling.

All the rock is broken in one operation and can be withdrawn steadily from the draw points. Thus, slusher moves are few. Pillar recovery, by the above discussed method offers maximum personal safety while drilling and the safest possible stope back after blasting. A horizontal ring blast leaves a clean back in pillar recovery at the Torbrit mine.

These factors were the primary objects in adopting longhole drilling to pillar recovery. The management feels that these original aims have been successfully met. In all, Torbrit has been satisfied with this type of ore extraction and type of drilling.

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Facts and Figures on the Atlas Shaft Sinking Project

by William H. Love

Atlas makes full use of up-to-date methods in sinking. Rock bolting, mucking machine, and attention to details help provide good record despite unfavorable ground.

ATLAS shaft was sunk to explore at depth a portion of the eastern end of the dry belt, or silver belt, of the Coeur d'Alene mining district. The Atlas mine is a mile east of Mullan and is on the south side of the Osburn fault. Equal partners in the venture are Hecla Mining Co., Newmont Mining Co., New Jersey Zinc Co. and Atlas Mining Co., the financing being equally shared by Hecla, Newmont, and New Jersey Zinc.

As outlined the project had as its goal deep exploration of prominent structures developed in the Atlas in upper levels. Plans called for deepening the shaft from the 800 level to the 2400 level with subsequent drifting, crosscutting, and diamond drilling on the 2400.

Preliminary work required prior to shaft sinking included rebuilding of the surface plant, improvement of the Atlas tunnel, installation of haulage facilities, cutting of a hoist room and rope raise, extending the shaft raise, installation of hoist, renovation of compressor plant, dewatering, and shaft preparation.

Surface Plant

The only old buildings suitable for use were one housing the compressor, and a small warehouse. The remaining old buildings were torn down and salvaged. A new building 32x50 ft was erected to house a dry, office, combined lamp room and stock room, and a small shop.

The first 1000 ft of the Atlas tunnel, originally driven by handminers, was on a steep grade and extremely small. To use mechanical haulage, it was necessary to enlarge this section, and it seemed expedient to also raise the grade to conform with the rest of the tunnel. A new portal was excavated and timbered 4 ft above the old one. New track was laid in the remainder of the tunnel to the top station, 9200 feet from the portal. Thirty-lb rail was used, and 5x6-in. ties laid on 24-in. centers. The first 5600 ft of the tunnel is straight, but the last 3500 ft is crooked and required sideswiping.

Diesel Haulage

For haulage on the Atlas tunnel level, 15 rocker dump type cars are used. These Coeur d'Alene Hardware & Foundry Co. cars have a capacity of 40 cu ft each. The length of haul, 9200 ft, made battery motor haulage uneconomical, and the crooked tunnel made a trolley installation impractical. Hence, a decision to try diesel haulage. A Plymouth diesel-hydraulic locomotive was delivered in October 1951 and has been in service since then. It has now been operated 3600 hr, or about 210,000 ton-miles. Performance of the diesel has been excellent, and its application in the Atlas tunnel is a natural.

About 25,000 cfm of air circulate in the tunnel. In addition to forced mechanical ventilation, four other precautions have made the use of diesel haulage underground successful. These are 1) use of fuel stops in the injection system to limit fuel injected at any engine rpm to 80 pct of ideal fuel-air ratio; 2) use of a premium, low-sulphur diesel fuel, sold by Union Oil Co.; 3) use of a torque converter in the power line to eliminate lugging, and 4) periodic changing of engine fuel injectors, plus frequent maintenance checks.

Hoisting

Following completion of haulage facilities, a 30x35-ft hoist room was cut. The shaft was raised 16 ft to bring the distance from turnsheet to sheave center-line to 114 ft. The rope raise averages 5x8 ft in section, is 170 ft long, and was driven at at angle of 38°.

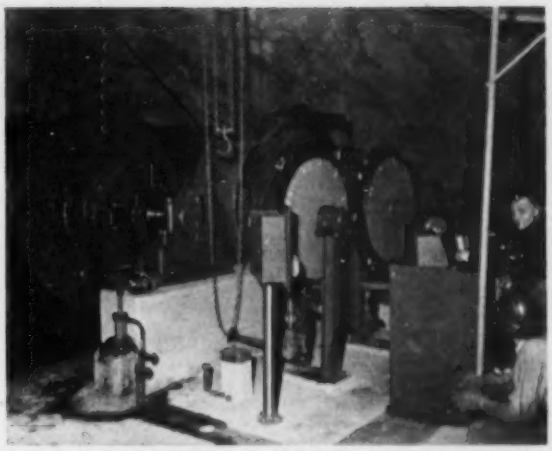
The hoist, manufactured locally by Coeur d'Alene Hardware & Foundry Co., is a tandem-drum machine with two 200 hp, 700 rpm motors. Both drums are overwind and this accounts for the small rope raise section. Brakes and clutches are controlled hydraulically. Rope speed is 1500 fpm, the sheaves are 5½ ft pitch diam, and the two 1-in. hoisting ropes are lubricated automatically as they pass over the sheaves.

A 125 hp Imperial type 10 compressor is installed underground at top station and furnishes the air needed by the operation. A CP 315 semiportable machine was installed at the Atlas and is used as an auxiliary.

Dewatering and Shaft Preparation

The Atlas shaft had been sunk to 800 ft by the Hecla Mining Co. in 1941 but this had been allowed to fill with water. Shaft dewatering was accomplished in about two weeks, using Ingersoll-Rand 35 sump pumps and 1½ MRV 30 motor pumps.

W. H. LOVE is Superintendent of the Hecla-Atlas project, Mullan, Idaho.



LEFT: Performance of diesel haulage has measured up to expectations. Plymouth 5-ton diesel locomotive is shown pulling a string of 40-cu ft rocker dump cars from Atlas tunnel portal. Haul is 9200 ft. RIGHT: Underground hoist is 400-hp tandem drum unit built by Coeur d'Alene Hardware & Foundry. Speed for the two 1-in. hoisting ropes is 1500 fpm, operating off 5½-ft sheaves.

Pump stations were installed on the 400 and 800 levels.

During the previous sinking, a single-drum hoist was used, and the outside compartment of the three-compartment shaft had been lined with 1-in. cedar for ventilation purposes. This lining was torn out, and guides were installed in the compartment.

The shaft was cleaned down and the manway repaired where necessary. It is interesting to note that no major repair work in the shaft was required as the timber was in almost perfect condition and no ground movement had occurred. A 4-in. air column, a 4-in. pump column, a 2-in. water supply line, and a 19-in. ventilation line were installed in the manway.

The Atlas shaft is three-compartment, each compartment 5 ft 6 in. x 4 ft 4 in. inside. Wallplates and endplates are 10x10 in., dividers 7½x12 in., S2S, making outside timber dimension 16 ft x 7 ft 2 in. Allowing a minimum of 12 in. for blocking, the rock section is not less than 9x18 ft, or 162 cu ft of rock per foot of depth. Actual broken muck hoisted per foot of advance was 290 cu ft. Sets were placed on 5-ft 3-in. centers, guides are 4x6 in., 21 ft long. Fir guides of satisfactory quality were not available at the time of preparation so Philippine mahogany was used.

Shaft Sinking Cost Per Ft

Direct labor	$68.00
Bits and rods	7.00
Compressed air	0.90
Drill repair	4.40
Explosives	9.25
Hoisting	15.80
Haulage	4.00
Pumping	3.25
Shaft mucker maintenance and royalty	3.00
Miscellaneous	4.00
Timber, pipe, etc.	40.00
Dump and surface	7.00
Power	3.90
Overhead—including insurance, general expense, taxes, social security, workmen's compensation, and payroll expense	10.50
	$181.00

To this figure of $181.00 must be added the cost of equipment, including drills, battleship, skips and cages, shaft-mucking machine, fans, and miscellaneous tools and equipment.

Shaft Sinking Crew

Sinking was on a three-shift, six-day basis, without Sunday or holiday work. The bottom crew consisted of four men, one a leadman. A topman was employed on day shift only, to see that timber was kept in readiness at top station, to make primers, and to see that all the various tools and supplies needed in the operation were kept in readiness. A mechanic was employed on day shift to maintain and repair the equipment involved. A motorman on the main haulage and two men outside, all on day shift only, completed the crew. The required electrical work was done out of the central Hecla shop at Burke.

Drilling and Blasting

Benching was used in advancing the shaft bottom. Full bottom rounds were tried and were successful, but benching was more advantageous. This was due to two factors: it was not possible to break the middle down to provide a sump, and the bottom was difficult to clean with holes collared in water and loose muck. Benching eliminated these disadvantages. The obvious disadvantages of benching are, of course, that it doubles set-up time and doubles smoke time.

Rock is lower St. Regis quartzite, which dips 80 to 85° to the north. Drill penetration rates varied from 8 to 18 in. per min as the ground varied in hardness. Ingersoll-Rand DA-30 drills were used throughout, mounted on a 3-in. pneumatic bar, which was set up parallel to and about 6 in. in from the endplate.

Carbide bits and rods were used during the early stages of sinking but were later replaced by Coromat 1-in. hex drill steel with a chisel-type insert. This steel gave faster drill penetration, was easier to handle, and improved bit and rod cost materially. The cost for bits and rods was slightly over 8¢ per ft of hole drilled.

Each hole was collared with a 2⅜-in. Carset bit, to receive a short length of 2-in. pipe casing. Due to the nature of the ground, it was found necessary to case nearly every hole. The casing was partially

String of empties and a timber car entering the Atlas tunnel. This Plymouth diesel had been in service 3600 hr, or 210,000 ton-miles. Battery or trolley locomotive use was ruled out, respectively, by length of haul and crooked tunnel.

recovered at three points—in the bottom, at the loading chute, and on the dump.

The round was 24, 27 or 30 holes, varying with the nature of the ground. Six rows of holes were drilled, 5 rows in front of the bar and 1 row behind. A 7½-ft round was drilled, and average rounds pulled just over 6 ft on the bench, or about 3 ft of full-shaft bottom. As soon as setting up was completed, two of the men left the bottom and did dead work such as fan lines and shaft lacing.

With the exception of the primer stick, which was 60 pct gelatin, the round was loaded with Du Pont 1⅛x8 Gelex No. 2 45 pct powder. All firing was done electrically from top station and Atlas regular delays were used. Millisecond delays were tried but no improvement in fragmentation was noted, and they also had a tendency to loosen blocking.

Ventilation was accomplished by two Joy 20-hp Axivane fans sucking through a 19-in. galvanized duct. A 3-hp Joy fan was installed above the lower end of the suction line and was used with a 12-in. fan bag to blow out the bottom and smoke-blowing time was reduced to about 10 min.

Mucking

Mucking was done with a Riddell clamshell mucking machine operated by one man. The carriage operated on a track which sat on the endplates of the second or third set from the bottom. The Blaw-Knox ⅜-cu yd clam proved very good in the rough muck in which it was necessary to operate.

During the mucking cycle, one man operated the clamshell, one sat on a wallplate at the end of the guides and rang bells, and two men stayed in the bottom. Mucking was into a 24-cu ft self-dumping skip, suspended on chains 33 ft below the sinking cage. All hoisting during mucking was done in counterbalance. Through the use of skips and a conventional overturn skip dump, a dump man was not required.

The two men in the bottom took turns tagging the clam, and did the barring and picking down of the walls. As soon as the bench was exposed, they cleaned it off so that when the last skip of muck went on the guides they were ready to set up to drill.

An all-steel battleship to protect the bottom set from damage during blasting and also to serve as staging during the timbering operation was lowered by the main hoist and suspended 7 ft below the bottom set by four chains hooked to the four ⅞-in. diam hanging rods. The staging floor was of 7-ft lagging. The two wallplates and two endplates were then lowered and hung. After the eight posts were installed, the set was pulled up tight and blocked in. Because of the use of the shaft mucker, dividers were left out of the bottom three sets.

Timbering

During the first part of the sinking, timber was put in two sets at a time. Bad ground conditions made this impossible later, however, and timber was carried within about 7 ft of the bottom.

The mucking machine and mucker rails were lowered as a unit by the main hoist. The mucker was moved down once for each two sets of advance. A set of half-length extension guides was used every other move. Panel lacing was used between sets, and was put in during the drilling cycle.

Stations were cut at 400-ft intervals—at the 1200, 1600, 2000, and 2400 levels. Motor pumps and tank sumps were installed on each level. Water from the bottom was pumped by Ingersoll-Rand 35 sump pumps, then relayed to the station pump by a 1½ MRV 30 motor pump installed on a 500-gal steel tank, lowered as the bottom advanced.

Rock Bolting

Use of rock bolts was of great value during the sinking, particularly in the last 200 ft where the ground became quite bad. In this section, a pattern calling for 12 rock bolts around the shaft every 5 ft was used. It is certain that use of the clamshell mucker would not have been possible without bolting because it would not have been possible to leave out dividers.

Above each station and before cutting, the shaft was bolted with particularly close attention paid to the brow. This proved successful.

Bolts were placed by the shaft crew as part of the contract. The 1-in. split-wedge type was used with ⅜x8x8-in. plates, and bolts were tightened with an impact wrench. Rock bolting contributed greatly to the safety of the men working in the bottom and eliminated serious sloughing. It is anticipated that it will also considerably reduce the amount of shaft repair work which would normally be expected.

Electrical

Joy connectors were used on all electrical equipment, including fans, pumps, and blasting cable. This made it unnecessary for the electricians to do such routine jobs as changing a pump or fan. This work was done by the shaft miners, with the help of the mechanic where required. The shaftmen also were able to extend the blasting cable and 440-v feeder for the bottom pump and auxiliary fan. Only one connector failure was experienced.

Sinking Record

The shaft was sunk and the four stations cut over a period of about 11 months, in 270 working days. The timbered advance was 1661 ft at an average advance of 6.3 ft per day.

Contract price to the 12 members of the shaft crew was $60 per ft. The three hoistmen and one topman shared in a small bonus, proportionate to the shaftmen's earnings. Average advance per man shift was 0.542 ft, and 14.76 direct man-hr were expended per foot of advance.

Development and Construction Of A Ground-Water Supply

by Owen F. Jensen, Jr.

- Geology
- Hydrology
- Chemistry
- Drilling
- Designing
- Recording

CONSTRUCTION of a ground-water supply includes many operations, which do not end with completion of facilities. Evaluations must be made of the quality of water in various areas and the history of production. Following an analysis of geologic, hydrologic, and chemical data there must be an exploratory drilling program and a study of resulting information. When the well is completed a continuous survey of operating records and basic data is necessary, since designs for future wells are varied as more data becomes available for those already completed.

As the following discussion must be limited to a single undertaking, the customary procedure will be outlined for development and construction of a ground-water supply in which more than three wells are involved. Parts of this procedure are applicable for all ground-water supplies.

Three possible sites were selected for construction of a paper mill. The problem was that the industry daily required approximately 15 million gallons of water of the best quality available; however, if water of the best quality were not available in the desired amount, water of a poorer quality could be used up to half the required amount, about 7 million gallons.

In the preliminary survey approximately 1500 to 2000 square miles were covered for each site. Collected data consisted of several hundred electric logs of oil and gas wells; reports by the U. S. Geological Survey on ground-water resources of localities in and near the area; unpublished records of several hundred water wells including chemical analyses of the water produced and drillers' logs of the wells; records of periodic water-level measurements made in observation wells by the USGS; topographic maps drawn up by USGS and by army engineers; and highway and county road maps. A preliminary report based on study and evaluation of these data was presented to the client, recommending the site with the most favorable and economical ground-water conditions. The study indicated that water of the quality and quantity desired could be developed. In this particular area there were indications that the aquifer could be separated into two zones containing water of different chemical character. This chemical character varied somewhat with area, but principally with depth, a factor which later proved an economic advantage in design of the well field.

After studying the preliminary report, the client weighed other economic factors as well as ground-water advantages and selected a plant site. Authority was given to proceed with detailed study of the chosen area.

Evaluation of preliminary data concerning the area indicated that it should be supplemented with additional information derived from actual exploratory drilling and testing. A flexible plan of exploratory drilling was devised so that data obtained as each test hole was drilled could be correlated with existing data and additional test holes could be located and drilled accordingly. Specific data to be obtained from each test hole were: samples of all formations penetrated; the driller's descriptive log of formations; electric logging surveys; one or more samples of water from selected water-bearing formations; temperature; and water levels and their recovery after periods of production.

The area studied in such detail was approximately 400 square miles. Choice of location for the original test holes was based on the preliminary study. The pattern for these test holes was designed to require a minimum number of holes and to allow for interpolation of data between two, three, and four holes. Geologic, hydrologic, and chemical data gathered after completion of the first four test holes showed that the north section was the most desirable in the area, which was then reduced from the original 400 square miles to about 150 square miles for more detailed and intensive study.

On the basis of findings from the current test a tentative well field was laid out and a drilling program devised. One test hole was located on the actual plant site and later developed into a small pilot production well equipped with a deep well turbine pump. Water from this well was used for general construction requirements.

Another test hole drilled near the pilot production well and adjacent to the proposed route of the well-collection system was used during the pumping test of the pilot production well to determine the degree of interference. From the data thus obtained, calculations were made to determine the coefficients of transmissibility and storage. Three additional holes

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were drilled along the proposed water-collection system. The completed exploratory test-drilling program consisted of eight test holes, five of which, including the pilot production well, were selected for completion into production wells. The remaining three were not completed because of their location relative to final plans for the collection line.

During exploratory testing and drilling the character and texture of the formations were determined by visual examination. Mechanical sieve analyses of samples of water-bearing material gave the grain size, grain-size distribution, uniformity coefficients, and effective sizes. Electric log surveys and drillers' logs indicated depth, thickness, and character of the water-bearing sands. As each water sample was taken from a formation short pumping and recovery tests were made. Together with two pumping and recovery tests made on the pilot production well and the observation well, these gave sufficient data. Variations in temperature and chemical quality of the water at different depths were ascertained; it was found that mineralization increased with depth. The chemical quality of water resulting from the mixing of different waters was calculated. Both specific capacity and drawdown and theoretical drawdown of the pressure level at the pilot production well were determined, and effects on artesian pressure in the observation well due to pumping of the pilot production well were noted. The coefficients of transmissibility and storage at different horizons in the aquifer were recorded.

On the basis of this data a report was prepared for the client. It was stated that the existing water-bearing formation was highly productive and could be depended upon to yield the desired quantity of water at economical levels. Although 60 to 70 million gallons were being withdrawn from this formation daily, it still had ample capacity to produce another 30 to 40 million gallons per day. Quality of the water, even at greater depths, was entirely satisfactory for the requirements outlined.

It was recommended that eight wells be spaced along the collection line at intervals of 2000 ft, seven to operate at 1600 gpm and the eighth to be held in reserve. The wells should be completed at two depths, approximately 750 and 1200 ft, and should alternate in depth. It was calculated that after five years of operation pumping levels would be 200 ft in the 750-ft wells and 350 ft in the 1200-ft wells. It was reported that after five years the levels would stabilize and remain essentially constant unless rate of withdrawal was changed. Natural recharge to the aquifer would be sufficient to supply the 15 million gallons pumped daily, in addition to other known and anticipated withdrawals. Additional areas adjacent to the proposed plant site could be developed to produce water for anticipated requirements.

After the receipt and acceptance of this report the plant site was selected and a final layout made of the well field. Designs for both the shallow and deep production wells included: 1—diameter and length of pump pit; 2—the most economical and efficient pump bowl diameters; 3—diameter and size of conductor casing; 3—type, diameter, size, and length of screen to be installed opposite water-bearing sands; 5—methods for developing the well; and 6—a specific testing procedure to obtain and evaluate hydrologic factors for each well so that preliminary estimates might be amplified and verified. The contract for drilling the production wells was drawn up, bids were taken, and the contract was awarded.

As each well was completed and tested, minor changes were made in the basic design as to gage of screen and size of gravel. Upon completion of each production well pumping tests were made to obtain hydrologic coefficients of the aquifer for the well field. Data from the tests were used to compute future pumping levels originally estimated on data available at the time of the preliminary survey. These hydrologic conditions were considered when pumping equipment was designed. Design, plans, and specifications were drawn and written for the pumping equipment, bids were taken, and a contract was awarded for installation of the deep well turbine pumps.

After all production wells were completed and the deep well turbine pumps were installed, a pumping procedure was set out for a series of production tests of all units before the wells were put into operation for plant processing. Data obtained from the tests and from chemical analysis of water from each well were used to determine the proper production rate for individual wells. On the basis of the testing procedure operating schedules during fluctuations in water demand were also recommended for each well. A rotation plan for using the standby well was outlined.

As in any ground-water development, withdrawals from wells are always accompanied by decline in water level. It is a basic concept that water levels must be depressed to cause water to flow toward the wells. In the well field described in this paper, the predicted decline in water levels meant only an increase in the pumping levels, and not a depletion of the water-bearing sands. However, to determine the accuracy of the predictions, which are based on pumping tests of comparatively short duration, basic data from the wells under normal operation should be collected. These data should include records of total pumpage from the well field, rates of production from individual wells, and pumping levels in the production wells. An analysis of the data in the light of predictions made of future water levels based on the hydrologic factors should be made periodically so the client can be kept informed of current ground-water conditions in the well field. Periodic samples of water from the wells should be collected so that any variation in chemical character of the ground water can be detected.

Unfortunately the described procedure is not yet generally accepted. In many instances specifications, locations, and depth and design of wells have been left entirely to the water-well contractor. Some industries, such as paper mills, mining companies, chemical plants, and power plants, frequently employ the services of consulting hydrologists. On the whole, however, only a small percentage of industries and municipalities have used the techniques of ground-water hydrology in planning and developing their well fields. Furthermore, plans for future expansions of well fields with respect to spacings, individual well yield, and other basic factors are lacking, or only casually considered in the general engineering for the plant. It is only through the application of ground-water hydrology to ground-water supplies that the proper use, development, and conservation of water can be realized.

Acknowledgments

Thanks are expressed to Nicholas A. Rose, who worked with the writer in preparing and supplying basic data for the paper, and to C. K. Hoffmeister, who reviewed and edited parts of the paper.

Structure and Mineralization at Silver Bell, Ariz.

by Kenyon E. Richard and James H. Courtright

SILVER Bell is situated 35 airline miles northwest of Tucson, Ariz., in a small, rugged range rising above the extensive alluvial plains of this desert region. Its geographical relation to other porphyry copper deposits of the Southwest is shown on the inset map in the lower left corner of Fig. 1. The climate is semi-arid. Altitudes range within 2000 and 4000 ft.

Opening of the Boot mine, later known as the Mammoth, in 1865 was the first event of note in the district's history. Oxidized copper ores containing minor silver-lead values were mined from replacement deposits in garnetized limestone and treated in local smelters. Copper production had approached 45 million pounds by 1909 when the disseminated copper possibilities in igneous rocks were recognized. Extensive churn drill exploration carried out during the next three years resulted in partial delineation of two copper sulphide deposits, the Oxide and El Tiro. Although the then submarginal tenor discouraged exploitation of these disseminated deposits, selective mining of orebodies in the sedimentary rocks continued intermittently until 1930, providing a production total of about 100 million pounds of copper.

The American Smelting & Refining Co. began exploratory and check drilling in 1948 and subsequently made plans for mining and milling the Oxide and El Tiro orebodies at the rate of 7500 tons per day. Production began in 1954 at a rate of about 18,000 tons of copper annually.

Formations ranging in age from Pre-Cambrian to Recent are exposed in the Silver Bell vicinity. The more erosion-resistant of these, Paleozoic limestone and Tertiary volcanics, predominate in the scattered peaks and ridges comprising the Silver Bell mountains. The porphyry copper deposits are located along the southwest flank of these mountains in hydrothermally altered igneous rocks. These are principally intrusives which cut Cretaceous and older sediments and are considered to be components of the Laramide Revolution.

For three-fourths of its length the zone of alteration strikes west-northwest, Fig. 1. There now is no single structure that accounts for this alignment. However, indirect evidence suggests that a fault representing a line of profound structural weakness existed in this position prior to the advent of Laramide intrusive activity. This line will be referred to as the *major structure*. It was obliterated by the Laramide intrusive bodies but exerted a degree of control on their emplacement, as evidenced by their shapes and positions. The influence of fault structures on the shapes of intrusives in other porphyry copper districts has been noted by Butler and Wilson[1] and by others.

As shown on the inset map on Fig. 2, a fault of parallel trend and considerable displacement lies to the north. This fault is now marked by a line of small Laramide intrusive bodies. To the south is a third fault of large displacement. Evidence of its age in relation to the Laramide intrusions and mineralization is not recognized, but its conformance in strike with the other two major faults is significant. These three breaks establish a pronounced trend of regional faulting. They are high-angle, and the southerly one may be reverse. Stratigraphic separations on these faults are of the order of several thousand feet.

The local Paleozoic section is about 4000 ft thick. It is composed predominantly of limestone with a basal quartzite member. The Cretaceous section appears to exceed 5000 ft. Conglomerates, red shales, and arkosic sandstones (the youngest) characterize the three principal members.

Intrusion of alaskite marked the beginning of Laramide igneous activity. It was emplaced as an elongate stock with one side closely conforming to the major structure line throughout a distance of nearly 4 miles. The alaskite was at one time regarded as a thrust block of pre-Cambrian rock[1]; however, its intrusive relationship and consequent post-Paleozoic age has been established by inclusions of limestone found in outcrops north of El Tiro.

The next event was the intrusion of a large stock of dacite porphyry into Paleozoic sediments and alaskite. The stock was some 3 miles wide and at least 6 miles long in a northwesterly direction. It was sharply confined along its southwest side by the major structure line. A number of large pendants of moderately folded Paleozoic sediments occur within and along its southwest edge. Thus the inferred, original major fault between Paleozoic and Cretaceous sediments became a contact between alaskite and Paleozoic sediments and then a contact between dacite porphyry and alaskite. Andesite porphyry may have been intruded later than the dacite porphyry, but relationships are not clear; it may be simpiy a facies of the latter.

The intrusive activity was at this stage interrupted by an interval of erosion. The erosion surface probably was rugged, as there were local accumulations of coarse, angular conglomerate. Subsequently a series of volcanic flows and pyroclastics several thousand feet thick was deposited. A similar unconformity has been recognized elsewhere in the Southwest, particularly in the Patagonia Mountains near the Flux mine some 75 miles southeasterly. Here, as at Silver Bell, volcanics were deposited on an erosion surface cut in Cretaceous and older sediments which had been intruded by alaskite.

Though no evidence is offered that closely defines the age of this unconformity, and proper analysis of the problem is beyond the scope of this paper, it is

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interesting to speculate that it may mark the close of the Cretaceous Period and provide a distinction between Tertiary and Cretaceous igneous rocks within the Laramide Revolution.

Subsequent parallel faulting along the major structure line sliced the volcanics and Cretaceous sediments into horst and graben structures. These faults are remarkably persistent southeasterly, extending several miles beyond the limits shown in the map. It is not clear whether or not they originally extended through the northwest part of the district; they may have terminated against the earlier east-west fault shown on the inset map in Fig. 2. The formation of these faults indicates at that time a still existent deep-seated zone of weakness along the major structure line.

Monzonite stocks and contemporaneous dikes were then emplaced along and near this line, obliterating portions of the faults described in the foregoing paragraph. The stocks are elongate parallel to the major structure line. The dikes are distributed along this line but trend across it, for the most part, with an average east-northeast strike. Systems of close-spaced parallel fractures then developed. Like the dikes, these fractures are distributed along the major structure and strike across it. Alteration and sulphide mineralization then took place. The deposition of sulphides, particularly chalcopyrite, was controlled in detail by the cross-trending fractures. Although these were distributed along the major structure line as a narrow band, it is notable that throughout much of its length there are now no fault structures to account for this trend.

Post-sulphide dikes of andesite represent the last intrusive activity in the immediate district. Curiously, most of these dikes parallel the strike of the major structure, although it would seem that the cross-breaking fractures represented available lines of weakness. This serves to emphasize the major structure line as being a profound, deep-seated zone of weakness persisting through a long period.

Uplift and erosion of the region during late Tertiary or Quaternary time exposed the lean primary mineralization to processes of weathering and enrichment, resulting in the accumulation of the two chalcocite orebodies.

Small plugs and flows of basaltic lava occur in the flats surrounding the Silver Bell range. These are later in age than the Gila conglomerate and mark one of the more recent events in the geologic history of the region.

Structural Control of Hypogene Mineralization

As in the majority of porphyry copper deposits, the principal primary sulphide minerals at Silver Bell are pyrite and chalcopyrite. Although occurring as disseminated grains, they are more abundant as narrow veins or seams which are usually near-vertical in attitude and persistently parallel. Varying in thickness from paper-thin to several inches and in spacing from inches to several feet, these thin sulphide sheets occur as groups of various sizes throughout the narrow northwest-trending zone of hydrothermal alteration. (Because of the small scale, a single line in the pattern of *Mineralized Fissures* on Fig. 1 usually represents a large number of parallel structures, rather than individual.) In detail the average individual fissure appears as a thin quartz-sulphide seam encased by a uniform band of sericite. The fissures are predominantly oriented in the northeast quadrant; a small proportion strike northwest and a few are random. From a broad viewpoint there are no intersections of consequence among these groups. Within groups changes in strike occur gradually and result in curving trends. As noted earlier, these systems of mineralized fissures are distributed along the major structure line, and it is assumed they were formed in response to deep-seated uniform stress related to this line.

At least a few hundredths of 1 pct copper is present nearly everywhere in the altered zone; better values occur where there are mineralized fissures and the best values where the fissures are close-spaced. The two comparatively large groups of these close-spaced structures coincide with the positions of the two orebodies, Fig. 1. However, the actual structural, mineralogical, and lithological distinctions among these and other, smaller groups are minor, and the factors that controlled the position and size of these two groups are not clearly evident. The strong east-west fault which terminates in the Oxide area may have influenced the concentration of fracturing there, and at El Tiro the sharp bend in the alteration zone and the group of northeast-striking dikes likewise may indicate a cross-trending line of weakness that localized stresses. Nonetheless, the importance of these structural conditions is not clearly demonstrated, and no good evidence is found to explain the structural cause of the more intense fracturing which localized the two orebodies in their present positions in preference to other locations along the major structure line.

Outside the zone of alteration the dacite porphyry is finely fractured and jointed throughout most of its large exposed area. In sharp contrast to the systems of parallel fissures in the alteration zone, these breaks in the dacite porphyry are almost completely disoriented; parallelisms are rare and traceable for only a few inches or feet. They appear to be premineral where they are found in the alteration zone in the westerly and southwesterly portions of the dacite porphyry. Evidently in physical aspect this formation was exceptionally well prepared to be mineralized, perhaps better than the rocks of the ore zone proper. The fact that it was mineralized only to a minor degree may be accounted for by the absence of systematic fractures. That is, only the systems of parallel fractures were connected with the deep-seated source of mineralization, and the pervasive breaking of the dacite porphyry did not alone qualify it for mineralization.

Except the post-mineral andesite dikes, all igneous rocks within the narrow northwest-trending zone shown on Fig. 1 are hydrothermally altered. Variations in the intensity or in the completeness of the process have been subdivided by Kerr[3] into five stages. His analysis demonstrated, among other things, that the known orebodies occur within the more strongly altered areas.

The area outlined on Fig. 1 includes all degrees of alteration, but no differentiation is made. It merely represents the areal extent of bleached-looking, igneous rocks showing evidence in the leached outcrops of pre-existing disseminated sulphides, principally pyrite. The transition to relatively fresh rock is sharp in many places, particularly along the contact with sedimentary rocks and on the faults in the southeast portion. However, along most of the southwest margin the transition is gradational, and the limit is an arbitrary line.

Tactite, composed essentially of garnet, quartz, and lime-silicate minerals, is confined to a narrow belt along the southwest margin of the limestone

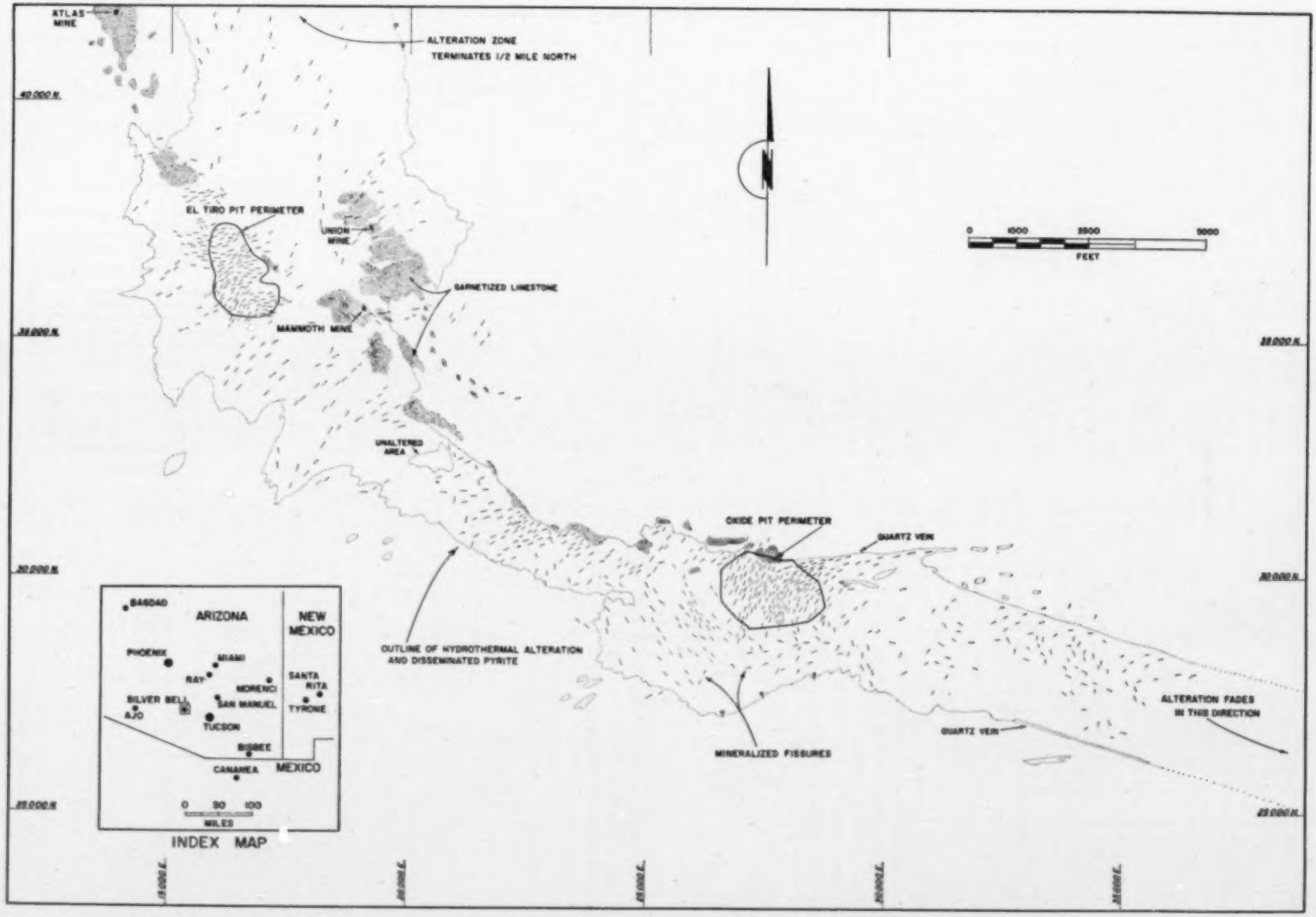

Fig. 1—Patterns of mineralized fissures, zone of hydrothermal alteration, and areas of pyrometasomatism in sedimentary rocks.

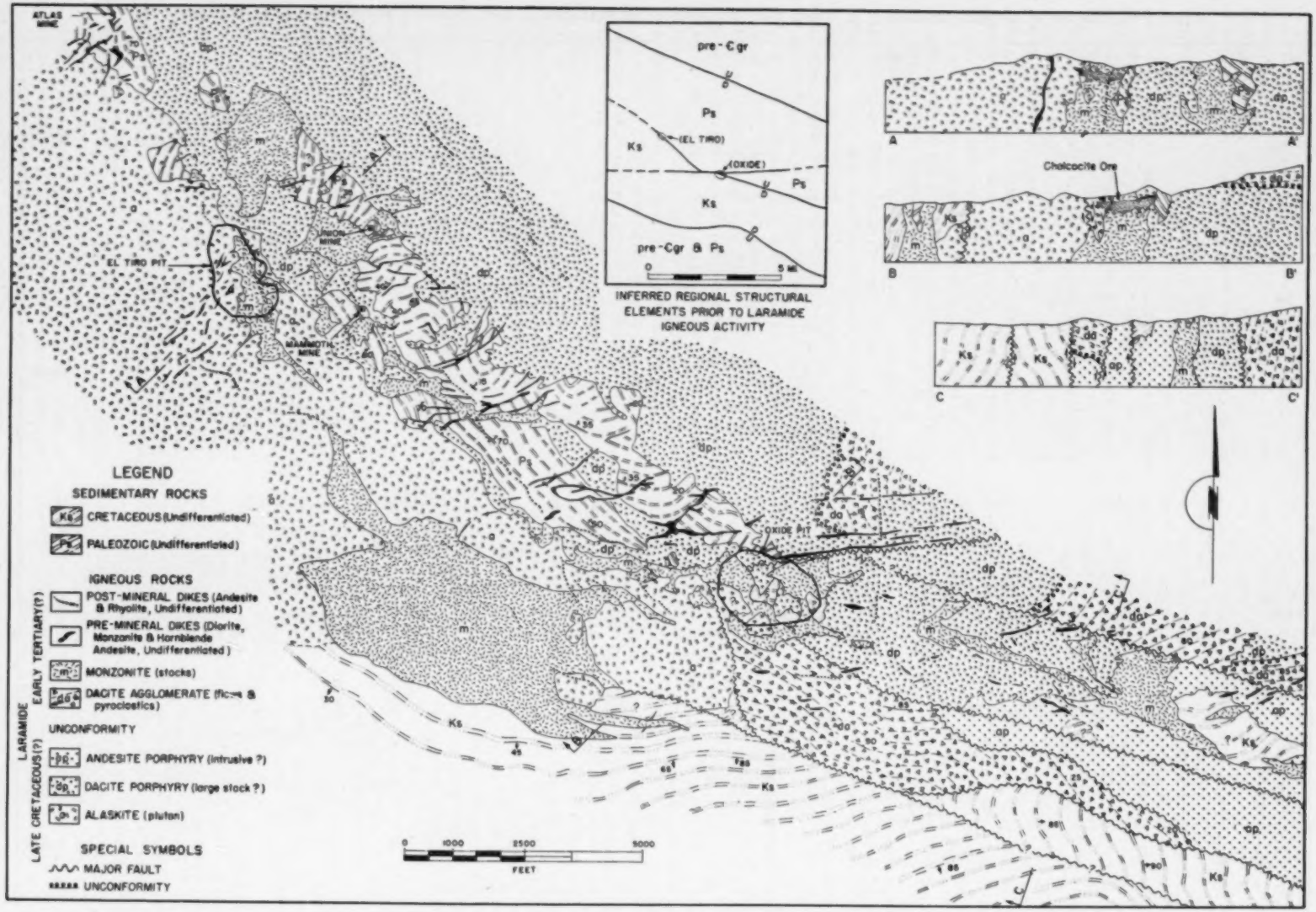

Fig. 2—Geologic map, Silver Bell district, Pima County, Ariz.

pendants, except in the vicinity of the Mammoth and Union mines where it has replaced the full width of the sedimentary block. It has been suggested by Stewart[3] that the dacite porphyry and monzonite are responsible for this *contact metamorphism*. Areal distribution of this tactite is such that, if it were to be considered strictly as a contact phenomenon, the alaskite would be as related to it as the other intrusives. Without going into the problem in detail, it is worth noting that the tactite occurs along the major structure line in such a manner as to indicate a close genetic connection with it. Supporting evidence in the form of well-defined structural controls of individual pods of tactite is not recognized. An occasional mineralized fissure cuts the tactite in the Mammoth and Union deposits, although the primary chalcopyrite orebodies have little obvious structural control. Elsewhere fissures in igneous rocks end abruptly at tactite margins.

Supergene Enrichment

The two ore deposits consist of rudely tabular accumulations of chalcocite from 100 to 200 ft thick. Lying beneath about 100 ft of leached capping, they were formed by two to threefold enrichment of the copper contained in the primary mineralization. Typical ore is composed of altered rock and sulphides in a ratio of about 10 to 1 by weight.

Most of the capping over the orebodies contains less than one-tenth of 1 pct copper as cuprite, or other oxidation products, mingled with the limonite. Occasionally somewhat higher values occur where copper has been precipitated as silicates and carbonates by reactive gangue material present in less altered rock. Within the orebodies, where alteration is strong and the gangue is nonreactive, the upper limit of the sulphide zone (the base of oxidation) appears on opencut faces as a sharply defined, highly irregular line. Only rarely is there a transition zone of mixed sulphide and oxidized minerals. In general shape the base of oxidation conforms to modern topograhy, even though local relief exceeds 200 ft. The water table for the most part now is well below the chalcocite zone.

Some of the irregularities of the base of oxidation are caused by displacement on post-chalcocite faults, but most are due to variations in rock permeability. This is evidenced by the dense siliceous character of a few sulphide remnants occurring well up in the leached zone, and by leached indentations of the sulphide zone along some of the fissures.

It is significant that although the base of oxidation shows general conformance to the topography, in detail it is a highly irregular, sharply defined front. Its present shape may have been produced by modification of a pre-existing base, one that was established during relatively moist climatic periods of the past. Under such conditions the water table would have oscillated at uniformly shallow depths, serving to limit the depth of oxidation and thereby determining the shape of its base to some extent. Otherwise, under conditions involving depression of the water table, the oxidation process apparently proceeded in the vadose zone independent of the water table, advancing downward on a sharply defined front whenever oxygen-charged meteoric water reached it.

Opinions vary both as to the role of the water table in the deposition of chalcocite and the reason for distribution of the chalcocite through a considerable vertical range. At Silver Bell pyrite and preferentially chalcopyrite are only partially replaced by chalcocite immediately below the line at the base of oxidation as well as on down through the zone of enrichment. This condition favors the theory that chalcocite is deposited at or below the water table. That is, the dissolved copper on its downward course bypassed available chalcopyrite and pyrite until it reached the water table where it formed chalcocite. The partial replacement of primary sulphides by chalcocite and its vertical distribution, as now existing, may then be explained as originating through the numerous cyclic fluctuations of the water table position.

Leached Outcrops

In the formation of most disseminated chalcocite deposits the enrichment process takes place progressively: copper is repeatedly dissolved, carried downward, and precipitated. It has been well established by Blanchard,[4] Locke[5] and others that under these conditions limonites of certain colors and textures are left behind in the leached capping as evidence of the pre-existing chalcocite.

The Silver Bell district provides exceptionally good examples of this phenomenon, but limonites of chalcocite derivation are not confined to the outcrops over the orebodies. They are widely dispersed through the zone of alteration. Proper interpretation of their significance in respect to ore possibilities rests mainly on quantitative rather than qualitative appraisal. Mapping of the Silver Bell outcrops on this basis provided a valuable guide in exploration drilling. Results have demonstrated conclusively that the pattern of relatively strong copper mineralization at depth is reflected in the outcrops by the distribution and abundance of diagnostic limonites.

Previous Work and Acknowledgments

The first scientific study of the district was published in 1912 by C. A. Stewart.[3] Considerable field and laboratory work has been done in more recent years by several groups and individuals, including the writers, all reporting privately to the American Smelting & Refining Co. Roland Blanchard conducted leached outcrop studies in a portion of the area. Harrison Schmitt with H. M. Kingsbury and L. P. Entwistle mapped structure and mineralization in the central part of the district. Paul F. Kerr studied the alteration features, and later published a comprehensive paper[2] on the district. Thomas Mitcham mapped structural features in the surrounding area. The writers have drawn considerably on unpublished data, particularly in compilation of the geologic map. The high quality and usefulness of the work of these men is gratefully acknowledged, but unfortunately it is not feasible to give specific individual credits.

Thanks are due the American Smelting & Refining Co. for permission to publish this paper.

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Discussion*

Contents

A — Metal Mining

Safety Factor Characteristic Curves. Their Application to Mine Hoisting Ropes. (Paper by W. A. Boyer, *Transactions* AIME, **199**, 989; *Mining Engineering*, October 1954. Discussion by B. E. Grant.) 1101

The Use of Wooden Rock Bolts in the Day Mines. (Paper by Rollin Farmin and Carville E. Sparks, *Transactions* AIME, **196**, 922; *Mining Engineering*, September 1953. Discussion by Edward Thomas.) 1101

An Analysis of Mine Opening Failure by Means of Models. (Paper by Bernard York and John J. Reed, *Transactions* AIME, **196**, 705; *Mining Engineering*, July 1953. Discussion by J. P. Zannaras.) 1102

B — Minerals Beneficiation

Flotation and the Gibbs Adsorption Equation. (Paper by P. L. de Bruyn, J. Th. Overbeek, and R. Schuhmann, Jr., *Transactions* AIME, **199**, 519; *Mining Engineering*, May 1954. Discussions by W. E. Ewers and Masayoshi Wada.) 1103

Manganese Upgrading at Three Kids Mine, Nevada. (Paper by S. J. McCarroll, *Transactions* AIME, **199**, 289; *Mining Engineering*, March 1954. Discussion by J. Bruce Clemmer, J. B. Rosenbaum, and C. H. Schack.) 1105

A Physical Explanation of the Empirical Laws of Comminution. (Paper by D. R. Walker and M. C. Shaw, *Transactions* AIME, **199**, 313; *Mining Engineering*, March 1954. Discussions by Dimitri Kececioglu, George B. Clark, and Fred C. Bond.) 1106

F — Coal

Underground Electrocarbonization of Coal and Related Hydrocarbons. (Paper by T. C. Cheasley, J. D. Forrester, and Erich Sarapuu, *Transactions* AIME, **199**, 908; *Mining Engineering*, September 1954. Discussions by John G. Tripp and James L. Elder.) 1108

Longwall Mining and Mechanization, with Special Reference to Nova Scotia. (Paper by Frank Doxey, *Transactions* AIME, **199**, 701; *Mining Engineering*, July 1954. Discussion by Louis A. Turnbull.) 1109

Cleaning Various Coals in a Drum-Type Dense-Medium Pilot Plant. (Paper by M. R. Geer, W. A. Olds, and H. F. Yancey, *Transactions* AIME, **196**, 696; *Mining Engineering*, July 1953. Discussion by John Griffen.) 1110

Frontiers in Heat Extraction from the Combustion Gases of Coal. (Paper by Elmer R. Kaiser, *Transactions* AIME, **199**, 304; *Mining Engineering*, March 1954. Discussion by G. A. Vissac.) 1112

Exploration of the Oaxaca Coal Fields in Southern Mexico. (Paper by Luis Toron and Salvador Cortes-Obregon, *Transactions* AIME, **199**, 505; *Mining Engineering*, May 1954. Discussion by John D. Price.) 1113

H — Industrial Minerals

Alkali Reactivity of Natural Aggregates in Western United States. (Paper by William Y. Holland and Roger H. Cook, *Transactions* AIME, **196**, 991; *Mining Engineering*, October 1953. Discussions by Dexter H. Reynolds and W. L. Merritt.) 1114

Characteristics of Titaniferous Concentrates. (Paper by L. E. Lynd, H. Sigurdson, C. H. North, and W. W. Anderson, *Transactions* AIME, **199**, 818; *Mining Engineering*, August 1954. Discussion by D. R. Grantham.) 1116

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* TP 3911

A — Metal Mining

Safety Factor Characteristic Curves. Their Application to Mine Hoisting Ropes.

by W. A. Boyer

DISCUSSION

B. E. Grant *(U.S. Smelting, Refining & Mining Co., Salt Lake City)*—There are several exceptions from the practical operating viewpoint that might be taken with the paper. The author recommends using a "curve" in plotting safety factors of ropes and suggests that the U. S. Bureau of Mines change its present "step curve" method to the "curve" method. From an engineering theoretical point of view that is possible, but from a practical standpoint it is not possible. Rope manufacturers fabricate their ropes in diameter gradients of 1/16 in. to 1/4 in. which automatically sets up a step pattern gradient. If ropes were manufactured in the diameter gradients of 1/1000 of an inch or thereabouts, the curve method could be used. Also, from a practical standpoint, if an operator is to hoist from a depth of 4000 ft, for example, he will determine the size of the rope necessary for that depth, and it is only natural that he will have safety factor to spare at lesser depths, and in plotting this excess capacity the natural end result is a curve. The U. S. Bureau of Mines tables are set up so that the operator will know what size rope at the required safety factor he will need at his maximum depth. He is not concerned with lesser depths.

W. A. Boyer *(author's reply)*—Mr. Grant states, "Rope manufacturers fabricate their ropes in diameter gradients of 1/16" to 1/4" which automatically sets up a step pattern gradient." There is no connection or relationship between the "size" pattern gradient and the present "factor of safety" step pattern gradient. The inherent factor of safety values for any particular size of rope with a given connected load when plotted for various depths is a *curve*, and with modern rope material and construction the inherent factor of safety curve is quite far removed from the present recommended factor of safety step curve. This is shown quite clearly in Figs. 3 and 4 of my paper.

Mr. Grant says, "If an operator is to hoist from a depth of 4000 ft, for example, he will determine the size of rope necessary for that depth, and it is only natural that he will have safety factor to spare at lesser depths." This is not true. I have prepared the paper with the prime purpose of showing the fallacy in only checking the factor of safety at the lowest depth and then trusting it is all right for the upper levels. I stressed in the paper the difference and showed in Fig. 4 that this difference is on the danger side when hoisting from levels above 4000 ft in the above-mentioned instance. With the installation calculated for a factor of safety of 4 at 4000 ft, static loading, a comparison for the upper levels is as follows:

Item	Depth of Hoisting, Ft				
	4000	2500	1500	750	250
Recommended Bureau of Mines values	4	5	6	7	8
Actual values considering static loading	4	4.53	4.96	5.35	5.64
If an acceleration rate of 3.216 ft per sec per sec is present the actual values would be	3.64	4.11	4.51	4.86	5.12

By far the simplest factor of safety rule would be to require that the factor of safety at 2500 ft equal 5.55. The inherent characteristic curve of the rope would then give a factor of safety at the various depths as indicated by my proposed curve. The characteristic curves for various size rope vary but slightly and for high rates of acceleration would allow a slight increase in factor of safety for the upper levels.

Use of Wooden Rock Bolts in the Day Mines

by Rollin Farmin and Carville Sparks

DISCUSSION

Edward Thomas *(U.S. Bureau of Mines, Washington, D. C.)*—This excellent article on an ingenious and successful installation of wooden rock bolts loses much of its effectiveness through an attempt by the authors to establish the general adaptability of this type of support. Possibly because they have only local experience as a guide, they imply erroneously in the first paragraph that the Bureau of Mines has been opposed to wooden bolting, although later in the article reference is made to a Bureau publication on the subject.

In 1947 the Bureau of Mines decided to sponsor experiments in roof bolting in coal mines because the experience of the St. Joseph Lead Co. had indicated that this type of roof support appeared to be a roof-fall accident-prevention measure that might be adopted and also effect economies in operating cost.

Justification for this belief has been borne out, particularly in coal mining. This program has been carried out under the following general program:

1—The Roof Control Section of the Bureau of Mines has advocated and initiated underground experiments under controlled conditions with many different ideas related to rock bolting since 1947. In no instance has cooperation in developing an idea been refused, no matter how ridiculous it may have seemed at first glance, if it has been requested by a mining company.

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Discussion*

Contents

A — Metal Mining

Safety Factor Characteristic Curves. Their Application to Mine Hoisting Ropes. (Paper by W. A. Boyer, *Transactions* AIME, **199**, 989; *Mining Engineering*, October 1954. Discussion by B. E. Grant.) 1101

The Use of Wooden Rock Bolts in the Day Mines. (Paper by Rollin Farmin and Carville E. Sparks, *Transactions* AIME, **196**, 922; *Mining Engineering*, September 1953. Discussion by Edward Thomas.) 1101

An Analysis of Mine Opening Failure by Means of Models. (Paper by Bernard York and John J. Reed, *Transactions* AIME, **196**, 705; *Mining Engineering*, July 1953. Discussion by J. P. Zannaras.) 1102

B — Minerals Beneficiation

Flotation and the Gibbs Adsorption Equation. (Paper by P. L. de Bruyn, J. Th. Overbeek, and R. Schuhmann, Jr., *Transactions* AIME, **199**, 519; *Mining Engineering*, May 1954. Discussions by W. E. Ewers and Masayoshi Wada.) 1103

Manganese Upgrading at Three Kids Mine, Nevada. (Paper by S. J. McCarroll, *Transactions* AIME, **199**, 289; *Mining Engineering*, March 1954. Discussion by J. Bruce Clemmer, J. B. Rosenbaum, and C. H. Schack.) 1105

A Physical Explanation of the Empirical Laws of Comminution. (Paper by D. R. Walker and M. C. Shaw, *Transactions* AIME, **199**, 313; *Mining Engineering*, March 1954. Discussions by Dimitri Kececioglu, George B. Clark, and Fred C. Bond.) 1106

F — Coal

Underground Electrocarbonization of Coal and Related Hydrocarbons. (Paper by T. C. Cheasley, J. D. Forrester, and Erich Sarapuu, *Transactions* AIME, **199**, 908; *Mining Engineering*, September 1954. Discussions by John G. Tripp and James L. Elder.) 1108

Longwall Mining and Mechanization, with Special Reference to Nova Scotia. (Paper by Frank Doxey, *Transactions* AIME, **199**, 701; *Mining Engineering*, July 1954. Discussion by Louis A. Turnbull.) 1109

Cleaning Various Coals in a Drum-Type Dense-Medium Pilot Plant. (Paper by M. R. Geer, W. A. Olds, and H. F. Yancey, *Transactions* AIME, **196**, 696; *Mining Engineering*, July 1953. Discussion by John Griffen.) 1110

Frontiers in Heat Extraction from the Combustion Gases of Coal. (Paper by Elmer R. Kaiser, *Transactions* AIME, **199**, 304; *Mining Engineering*, March 1954. Discussion by G. A. Vissac.) 1112

Exploration of the Oaxaca Coal Fields in Southern Mexico. (Paper by Luis Toron and Salvador Cortes-Obregon, *Transactions* AIME, **199**, 505; *Mining Engineering*, May 1954. Discussion by John D. Price.) 1113

H — Industrial Minerals

Alkali Reactivity of Natural Aggregates in Western United States. (Paper by William Y. Holland and Roger H. Cook, *Transactions* AIME, **196**, 991; *Mining Engineering*, October 1953. Discussions by Dexter H. Reynolds and W. L. Merritt.) 1114

Characteristics of Titaniferous Concentrates. (Paper by L. E. Lynd, H. Sigurdson, C. H. North, and W. W. Anderson, *Transactions* AIME, **199**, 818; *Mining Engineering*, August 1954. Discussion by D. R. Grantham.) 1116

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* TP 3911

A — Metal Mining

Safety Factor Characteristic Curves. Their Application to Mine Hoisting Ropes.

by W. A. Boyer

DISCUSSION

B. E. Grant *(U.S. Smelting, Refining & Mining Co., Salt Lake City)*—There are several exceptions from the practical operating viewpoint that might be taken with the paper. The author recommends using a "curve" in plotting safety factors of ropes and suggests that the U. S. Bureau of Mines change its present "step curve" method to the "curve" method. From an engineering theoretical point of view that is possible, but from a practical standpoint it is not possible. Rope manufacturers fabricate their ropes in diameter gradients of 1/16 in. to 1/4 in. which automatically sets up a step pattern gradient. If ropes were manufactured in the diameter gradients of 1/1000 of an inch or thereabouts, the curve method could be used. Also, from a practical standpoint, if an operator is to hoist from a depth of 4000 ft, for example, he will determine the size of the rope necessary for that depth, and it is only natural that he will have safety factor to spare at lesser depths, and in plotting this excess capacity the natural end result is a curve. The U. S. Bureau of Mines tables are set up so that the operator will know what size rope at the required safety factor he will need at his maximum depth. He is not concerned with lesser depths.

W. A. Boyer *(author's reply)*—Mr. Grant states, "Rope manufacturers fabricate their ropes in diameter gradients of 1/16" to 1/4" which automatically sets up a step pattern gradient." There is no connection or relationship between the "size" pattern gradient and the present "factor of safety" step pattern gradient. The inherent factor of safety values for any particular size of rope with a given connected load when plotted for various depths is a *curve*, and with modern rope material and construction the inherent factor of safety curve is quite far removed from the present recommended factor of safety step curve. This is shown quite clearly in Figs. 3 and 4 of my paper.

Mr. Grant says, "If an operator is to hoist from a depth of 4000 ft, for example, he will determine the size of rope necessary for that depth, and it is only natural that he will have safety factor to spare at lesser depths." This is not true. I have prepared the paper with the prime purpose of showing the fallacy in only checking the factor of safety at the lowest depth and then trusting it is all right for the upper levels. I stressed in the paper the difference and showed in Fig. 4 that this difference is on the danger side when hoisting from levels above 4000 ft in the above-mentioned instance. With the installation calculated for a factor of safety of 4 at 4000 ft, static loading, a comparison for the upper levels is as follows:

Item	Depth of Hoisting, Ft				
	4000	3500	1500	750	250
Recommended Bureau of Mines values	4	5	6	7	8
Actual values considering static loading	4	4.53	4.96	5.35	5.64
If an acceleration rate of 3.216 ft per sec per sec is present the actual values would be	3.04	4.11	4.51	4.86	5.13

By far the simplest factor of safety rule would be to require that the factor of safety at 2500 ft equal 5.55. The inherent characteristic curve of the rope would then give a factor of safety at the various depths as indicated by my proposed curve. The characteristic curves for various size rope vary but slightly and for high rates of acceleration would allow a slight increase in factor of safety for the upper levels.

Use of Wooden Rock Bolts in the Day Mines

by Rollin Farmin and Carville Sparks

DISCUSSION

Edward Thomas *(U.S. Bureau of Mines, Washington, D. C.)*—This excellent article on an ingenious and successful installation of wooden rock bolts loses much of its effectiveness through an attempt by the authors to establish the general adaptability of this type of support. Possibly because they have only local experience as a guide, they imply erroneously in the first paragraph that the Bureau of Mines has been opposed to wooden bolting, although later in the article reference is made to a Bureau publication on the subject.

In 1947 the Bureau of Mines decided to sponsor experiments in roof bolting in coal mines because the experience of the St. Joseph Lead Co. had indicated that this type of roof support appeared to be a roof-fall accident-prevention measure that might be adopted and also effect economies in operating cost.

Justification for this belief has been borne out, particularly in coal mining. This program has been carried out under the following general program:

1—The Roof Control Section of the Bureau of Mines has advocated and initiated underground experiments under controlled conditions with many different ideas related to rock bolting since 1947. In no instance has cooperation in developing an idea been refused, no matter how ridiculous it may have seemed at first glance, if it has been requested by a mining company.

2—The Bureau of Mines has no police powers over the selection of roof-support devices and seeks none.

3—At the request of an operator, the Bureau of Mines will make comparative tests on various types of bolts underground and will submit resultant data for the operator's consideration.

4—Wherever possible, the Bureau publicizes successful installations, so that others may benefit from this experience; however, it is careful not to discriminate between manufacturers of bolts or bolt-installation equipment. Unsuccessful experiments probably should also be publicized, but from a practical standpoint it is usually not feasible.

In other words, the role of the Roof Control Section of the Bureau is in line with the original purpose for which the Bureau was first organized: to serve industry through gathering, analyzing, and disseminating factual data. It would seem to be logical that the starting point for any experimentation on roof bolts in coal mines should be the time-proved practice of the

St. Joseph Lead Co. in sedimentary rocks; however, different sets of operating and economic conditions made it desirable and inevitable that many changes would be made during the adoption. At no time has the Bureau resisted these changes. It has exerted influence only to insure that experiments be carried on in a safe and orderly manner.

The first installation of wooden bolts was made in 1949 by Sterling S. Lanier, Jr., in the Rio Verde mine of the Norton Coal Corp., operating in western Kentucky, and the first publicity given this installation was at the 1949 National Safety Council in Chicago by Leon Kelly of the Roof Control Section of the Bureau of Mines, who had assisted Lanier in his experiments. Several coal mines in central Pennsylvania, northern West Virginia, and western Kentucky became interested in the Lanier bolt and have done considerable experimental and development work. It was not successful and interest lagged for a time. In 1952 G. R. Spindler of the Mining Dept., in cooperation with the Forest Products Laboratories of West Virginia University, started a test program in the Scotts Run field of northern West Virginia with a wooden bolt of new design. It is understood that the results of these tests are not yet ready for publication.

The Boone County Coal Corp., in cooperation with the Bureau of Mines, has been experimenting with wooden bolts in the Winding Gulf field of central West Virginia since 1949 and has not yet developed one that meets with the company's entire satisfaction, but a new wooden bolt and a new installation technique still offer promise of success, if they can be brought within a competitive cost range.

Therefore, contrary to the impression the authors seem to have, more widespread use of wooden bolts is not being and has not been hindered by opposition of the Bureau of Mines but through the inherent limitations imposed on this method by the materials and the techniques developed. Regardless of any state or federal governmental influence, considering the highly competitive coal market, anyone is indeed näive who thinks that a saving of $1.05 in the material cost of a bolt would not swing at least three-fourths of the coal mines to wooden bolting if it could be done safely.

It is believed that there is a definite place for wooden bolts in mining, and the article illustrates the ideal application: to reinforce the skin area of an arched opening and thus maintain the integrity of the arch by preventing sloughing of loose material through a maximum of bearing surface provided by the slabwood, and through the large amount of bearing surface between the wooden rod and the sides of the hole which provides shear strength to stiffen the structure. In such a formation, that is wet, loosely consolidated quartzite, application of tension at the bearing plate is unnecessary, and it would be impossible to maintain such tension if it were necessary.

Proponents of wooden bolts for beam building point out that many woods are comparable to steel in tensile strength parallel to the grain. However, no one has as yet suggested a practical way to take advantage of this strength. Pull tests with hydraulic jacks on such bolts show that only a small fraction of this strength can be utilized. Actually in the way these bolts are used, this test is not a criterion of effectiveness. If a wooden bolt is installed to maximum efficiency its maximum usefulness is in shear, not in compressing the rock by applying tension between a bearing plate and a fastening device inserted in the drilled hole, and wood can not resist shear efficiently, especially parallel to the grain. It is suggested that the back of the hole can be reamed, and by driving a wooden pin with a ram or a percussion tool obtain a much better bearing at the back than has been possible heretofore. This may well be so, but no practical bearing plate has been suggested that will take full advantage of this increased anchorage.

From investigative data gathered by the Bureau of Mines the following limitations prevail:

1—Wooden bolts have not been successful to date in compacting rocks in significant spans, except in marginal support cases, where approximately 90 pct of the rock would stand intact without any form of artificial support. The installation described in I. C. 7637 is of this nature.

2—It is unsafe to depend on the swelling action of the wood in a humid, wet atmosphere to expand the wood in the drilled hole, unless it is established that there is no seasonal humidity change in the mine. This is not a factor at the Day Mines.

3—The cost of drilling the large-diameter holes, plus the cost of adequate storage underground and on the surface, increased installation time, and other factors that are not immediately apparent, sometimes increase the installed cost of wooden bolts beyond the price of installed steel bolts.

An Analysis of Mine Opening Failure by Means of Models

by Bernard York and John J. Reed

DISCUSSION

J. P. Zannaras *(U.S. Tungsten Corp., Congress, Ariz.)* —In the investigation described in the paper by Bernard York and John J. Reed, the 8x8x8-in. square block was subjected to compressive forces from three sides, top, side, and rear.

Let us refer to the stresses set up by these forces as S_v due to top pressure, S_s due to side pressure, and S_r due to rear pressure.

At the section across the 2-in. hole (when failure occurred at 317 psi) the compressive stress S_v was equal to $317 \times 8/6 = -422$ psi, the top compressive stress S_v is of the same intensity, —422 psi, and the rear compressive stress S_r equals —317 psi.

The true stress is $T = S_v - \lambda S_s - \lambda S_r$.

Taking λ as 0.2,*

$$T = -422 + 0.2 \times 422 + 0.2 \times 317 = -274 \text{ psi.}$$

Taking the case when the opening was ½ in. and failure occurred at 517 psi,

$$S_v = 517 \times \frac{8}{7.5} = -537 \text{ psi}$$

$$S_s = 517 \times \frac{8}{7.5} = -537 \text{ psi}$$

$$S_r = -517 \text{ psi}$$

The true stress $T = S_v - \lambda S_s - \lambda S_r$

$$T = -537 + 0.2 \times 537 + 0.2 \times 517 = -327 \text{ psi.}$$

It can be shown that the above fundamental formula holds good not only for the above two examples but for all sizes of openings used, because when this formula is expressed in terms of the external pressure and size of the opening it agrees perfectly with the experimental results obtained by the authors and given in Fig. 12 of their paper.

General formula: $T = S_v - \lambda S_s - \lambda S_r$.

* See Mansfield Merriman's *Mechanics of Materials*, 11th ed., pp. 360 and 361.

Since all stresses are compressive the formula can be written for this particular case as follows:

$$T = -S_y + \lambda S_x + \lambda S_y \qquad \text{since } S_y = S_x$$

$$T = S_y(\lambda - 1) + \lambda S_x$$

$$S_y = \frac{P \times 8}{8 - V} = S_x$$

$$S_x = P$$

P = pressure, psi

V = size of opening in inches

T = true stress = strength of material = 300 psi

Substituting in the above formula, transporting, adding and subtracting similar terms we obtain $PV + 24P - 1500V = -12{,}000$, which is the equation of a hyperbola and shows the relation of pressure (causing failure) to the size of the opening.

Fig. 14 shows the experimental curve obtained by the authors in their Fig. 12 and the curve derived by the above equation.

It should be noted in the authors' experimental curve that as the size of the opening approaches 0 the stress becomes infinity, which is inconsistent, while the true stress formula shows that when the opening approaches 0 an external pressure of 500 psi will cause a true stress of 300 psi which will rupture the block.

Also it should be noted that if a diagram of stress was drawn against the size of the opening according to the authors it will follow the pressure curve, being twice its numerical value and varying from −364 psi to −1034 psi to infinity, which is out of the question, whereas in the true stress formula the stress induced to the material is always 300 psi, which is equal to the compressive strength of the material, and its graphic representation will be a straight line parallel to the X axis.

In view of the above facts it appears that the stress analysis of the authors must be in error and therefore their explanations for the so-called anomalous results they obtain seem unnecessary.

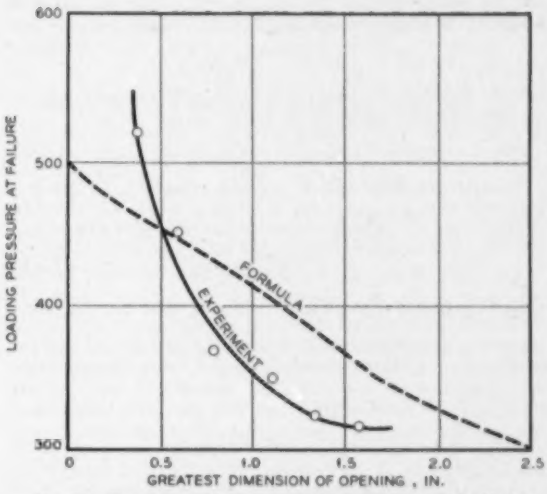

Fig. 14—Comparison of experimental curve obtained by York and Reed in Fig. 12 with curve derived from equation submitted by Zannaras.

It is obvious that the formula applied for the square block also applies for the cylindrical block.

John J. Reed *(author's reply)*—With regard to the above discussion, the *true stress* calculated by Mr. Zannaras is actually an average stress over the entire area of the block at the level of the opening. This takes no account of the known tendency for stress concentration about an opening in a solid subject to external stresses. I refer Mr. Zannaras to the U.S. Bureau of Mines Report of Investigations 4192, *Stress Analysis Applied to Underground Mining Problems*, for an excellent treatment of this subject. Also, we referred in our paper to *theoretically indicated* stresses at the boundaries of the openings, and thus far no attempt has been made to measure the actual stresses in the models. During the coming year I shall be continuing this study, and I plan to measure actual stresses experimentally and compare them with theoretical results. I hope to prepare a paper giving the findings of these studies when they are completed.

B — Minerals Beneficiation

Flotation and the Gibbs Adsorption Equation

by P. L. de Bruyn, J. Th. G. Overbeek, and R. Schuhmann, Jr.

DISCUSSION

W. E. Ewers *(Commonwealth Scientific and Industrial Research Organization, Melbourne, Australia)*:—Any attempt to elucidate further the meaning of the contact angle, particularly if it deals with the magnitudes of those mysterious quantities γ_{sA} and γ_{sL}, is certain to receive attention from those interested in the fundamentals of flotation. It is imperative, therefore, that attention should be drawn to a serious defect in the treatment by de Bruyn, Overbeek, and Schuhmann.

Their Eq. 5

$$d\gamma = -\Gamma_x d\mu_x - \Gamma_{H_2O} d\mu_{H_2O}$$

states correctly the relationship between the change in surface free energy $d\gamma$, the surface excesses Γ_x and Γ_{H_2O} of the components x and H_2O and the changes $d\mu_x$ and $d\mu_{H_2O}$ in the chemical potentials of these components. This equation applies for constant pressure and constant temperature and for the interface of the solid in *contact with the solution of X*. The relation remains valid when Γ'_x is defined so that Γ_{H_2O} becomes zero and the equation becomes

$$d\gamma = -\Gamma'_x d\mu_x \qquad [17]$$

From this equation de Bruyn et al. correctly concluded that increased adsorption of the collector at the liquid solid interface would reduce the interfacial energy γ_{sL}. Their second conclusion, however, that an increase in the contact angle, corresponding to a greater decrease in γ_{sA} than in γ_{sL}, can be attributed to greater adsorption of X at the solid-air interface than at the solid-liquid interface is quite unjustified.

The portion of the solid surface from which the collector solution has been displaced by air must be considered in terms of the equilibria between the gaseous phase and the surface. If the pressure and temperature

are constant, and if we assume that the collector is volatile, an equation identical to Eq. 5 applies

$$d\gamma_{sL} = -\Sigma\, \Gamma_i^A d\mu_i$$

which allowing similar assumptions to those made by de Bruyn et al. reduces to

$$d\gamma_{sL} = -\Gamma_s^A d\mu_s - \Gamma_{w,0}^A d\mu_{w,0} - \Gamma_y^A d\mu_y \qquad [18]$$

Here y indicates the new component, air, in the system and the superscripts, A, on the terms for the surface excess of collector and water indicate that these quantities are not necessarily identical with the corresponding terms in Eq. 5. We can be sure, indeed, that $\Gamma_{w,0}^A$ is different, because water will be positively adsorbed and $\Gamma_{w,0}^A$ cannot be assumed to be zero in the presence of air as the *solvent* component. It should be recognized that the chemical potentials here would normally be calculated from the partial pressures of the components but our assumption of equilibrium ensures that these potentials will be the same as in the other parts of the system and are determined therefore by the composition of the collector solution.

It will be noted, however, that in spite of this fact that changes in μ_i in the solution can be identified with those in the gaseous phase, the introduction of the terms $\Gamma_{w,0}^A$ and $\Gamma_y^A d\mu_y$ makes it impossible to compare Γ_s' and Γ_s^A. Thus the argument of de Bruyn et al. is deprived of its essential step. Their conclusion is therefore not proved, although one must admit the possibility that more collector may be in fact adsorbed per unit area at the solid-air interface than at the solid-liquid interface.

The main objection to their treatment of the system in which the collector is volatile applies equally well to the consideration of an involatile collector.

Essentially the theory of de Bruyn et al. fails because it does not account for the adsorption of water on the hydrophobic surface, and does, in fact, assume it to be zero.

Harkins et al.[m] have measured the free energy change (π_e) when dry graphite is equilibrated with saturated water vapor, and also the free energy change for complete emersion of the graphite $f_{s(SL)}$. These quantities for three samples of graphite each designated by its ash content are set out in Table I.

Table I. Free Energy of Adsorption of Water in Vapor π_e, and Free Energy of Emersion $f_{s(SL)}$, ergs cm^{-2}

Graphite, Ash Content, Pct	π_e	$f_{s(SL)}$
10	56	62
0.46	19	25
Less than 0.004	19	25

The equilibrium contact angle of the pure graphite measured under conditions designed to ensure that the surface had attained equilibrium with respect to the adsorption of water was 85.7°.

Here we see a considerable change in surface free energy (or surface tension) brought about by the adsorption of water vapor on a hydrophobic surface in a system uncomplicated by the adsorption of a collector. It will be evident that changes arising from the same cause must occur also on surfaces where collectors have been adsorbed to make them hydrophobic, and that it is certainly not permissible to ignore these changes in discussing the thermodynamics of the system.

[m] W. D. Harkins, G. Jura, and E. H. Loeser: Adsorbed Films of Water and Normal Heptane on the Surface of Graphite. *Journal of the American Chemical Society* (1946) 68, p. 554.

Masayoshi Wada *(Research Institute of Mineral Dressing and Metallurgy, Tohoku University, Nagamachi, Sendai, Japan)*—By combining Gibbs' equation with Szyszkovski's equation, two equations, Eqs. 19 and 20, expressing the time variation of surface tension due to the change in concentration caused by the preferential evaporation of one component of a solution were introduced according to the assumption that the rate of diffusion of molecules to the surface from the body of solution is small as compared with that of evaporation. It was verified experimentally that the second equation, in which the rate of diffusion of molecules to the surface from the body of solution is assumed to be small as compared with that of evaporation, held good in the aqueous solutions of cineol.

$$H_0 - H = bH_0 \ln (1 - c_i/(a + c_i) \cdot e^{-Kt}), \qquad [19]$$

$$H_0 - H = bH_0 \ln (c_i/a \cdot e^{-Kt} + 1), \qquad [20]$$

where H is the surface tension of solution, H_0 that of pure water, t the time, c_i the initial concentration, K the time constant, a the concentration coefficient, and b the surface tension coefficient.

The relation between the surface pressure F and the surface concentration u was expressed as follows:

$$F = -bH_0 \ln (1 - uRT/bH_0), \qquad [21]$$

where R is the gas constant and T the absolute temperature. It was verified that the constant b in Eq. 21 varied but little with frothers, and the differences in the surface activity of frothers were mainly due to the ease of adsorption of the molecules of frothers to the surface from the body of a solution.

Further, the relation between the surface pressure and the area S occupied by a gram molecule was expressed as follows:

$$FS/RT = Fe^F/bH_0 \,(e^F/bH_0 - 1). \qquad [22]$$

It was shown that the equation was applicable to the molecules displaying gaseous state in the adsorbed layer.

By combining Gibbs' equation with Langmuir's adsorption isotherm and Young's equation, Eq. 23 expressing the wetting isotherm in a solid-liquid-gas system was developed, which held good for the wetting of paraffin-wax in aqueous solutions of frothers.

$$H \cos x - H_0 \cos x_0 = b'RT \ln (c/a' + 1), \qquad [23]$$

where x is the angle of contact of solid against solution, x_0 that of solid against pure water, and a' and b' are constants. As the value of $H \cos x - H_0 \cos x_0$ corresponded to the surface pressure exerted by the adsorbed molecules at the solid-liquid interface, it was alternatively called the surface pressure of wetting, by which the wetting power of a surface-active substance was quantified. As the wetting isotherm, Eq. 23, was entirely similar in form to Szyszkovski's equation, the wetting activity and the orientation of frothers at paraffin-water interface was discussed as in the liquid-gas system.

A method capable of measuring directly the wetting tension of solid in a solid-liquid-gas system was developed and the wetting tension-time Eq. 24 was derived from Langmuir's equation of simple adsorption of gas on the surface of solids.

$$(B - B_0)/(B_1 - B_0) = q_1(1 - e^{-nt}), \qquad [24]$$

where B_0 is the wetting tension of the surface not covered by gas molecules, B_1 that of the surface perfectly covered by gas molecules, and B that corresponding to any fraction of the surface covered by gas molecules in a unit area of the surface, q_1 the fraction of the area covered by the adsorbed molecules at the adsorption equilibrium, and n the time constant. It was shown that the wetting tension of crystal glass at the water-air interface decreased fairly rapidly with time and was expressed fairly well by Eq. 24.

In the above discussion, the studies were limited to the simple cases, in which the molecules of pure frothers adsorbed at the water-air, paraffin-water-air, or glass-water-air interface. However, the studies should be extended to still more complicated cases encountered in usual flotation practices.

Thanks are due to Professor Takanori Mayeda, Kyushu University, for his support and criticism.

[23] Masayoshi Wada: *Journal of Mining Institute of Japan* (1944) 60, p. 109; (1949) 65, p. 253.
[24] Masayoshi Wada: *Journal of Mining Institute of Japan* (1943) 59, p. 681.
[25] Masayoshi Wada: *Journal of Mining Institute of Japan* (1947) 63, p. 71.
[26] Mayaoshi Wada: *Science Reports of the Research Institute of Tohoku University* Ser. A (1949) 1, p. 425.
[27] Masayoshi Wada: *Science Reports of the Research Institute of Tohoku University* Ser. A (1950) 2, p. 102.
[27] Masayoshi Wada: *Science Reports of the Research Institute of Tohoku University* Ser. A (1950) 2, p. 127.

P. L. de Bruyn, J. Th. G. Overbeek, and R. Schuhmann, Jr. *(authors' reply)*—Mr. Ewers' discussion is much appreciated, as it focuses attention on the problem of defining and measuring quantities adsorbed at the solid-gas interface. Also, Mr. Ewers calls attention to the experimental fact that H_2O is adsorbed on solid surfaces from the gas phase. This fact we recognize fully and took into account but did not discuss in detail in the paper. However, Mr. Ewers objects to 1—our use of the convention $\Gamma_{H_2O} = 0$ for the solid-gas surface and to 2—our conclusion that for simple collection systems there must be greater adsorption of the collector at the solid-gas interface than at the solid-water interface. We find these objections invalid, as shown below.

Mr. Ewers' Eq. 18 may be accepted as basis for discussion. The term $\Gamma_\gamma^A d\mu_\gamma$ is small and unimportant, and is, in fact, nil for flotation with an inert gas ($\Gamma_\gamma^A = 0$) or vacuum flotation (no γ-component present). In our analysis we have assumed that only the chemical potentials, μ_X and μ_{H_2O}, change by addition of the collector *X* to the aqueous phase. The chemical potential of *air* is assumed to remain constant. This is a good approximation provided the concentrations of the collector in the gas and the liquid phase remain small. Finally, if this term were significant, we would expect contact angles and mineral floatabilities to be sensitive to variations in air pressure, which is contrary to experiment.

Thus, Eq. 18 reduces to

$$d\gamma_{SA} = -\Gamma_X^A\, d\mu_X - \Gamma_{H_2O}^A\, d\mu_{H_2O} \qquad [25]$$

Adopting the convention that $\Gamma_{H_2O} = 0$, to which Mr. Ewers objects, Eq. 25 of course reduces to our Eq. 5, and we are led to the conclusion that Γ_X is greater at the solid-gas interface than at the solid-water interface. Our convention as applied to solid-gas interface will be understood more clearly if we proceed from Eq. 25 in a more general way. The Gibbs-Duhem relation for the solution phase, with all solutes at constant chemical potential except for the collector gives

$$c_X\, d\mu_X + d\mu_{H_2O} = 0 \qquad [26]$$

where c_X is the collector concentration in mols of *X* per mol H_2O. Solving for $d\mu_{H_2O}$ and substituting in Eq. 25

$$d\gamma_{SA} = -[\Gamma_X^A - c_X\,\Gamma_{H_2O}^A]\, d\mu_X \qquad [27]$$

If we define Γ_X as $(\Gamma_X^A - c_X\Gamma_{H_2O}^A)$, in which Γ_X^A and $\Gamma_{H_2O}^A$ are experimentally measured surface excesses in the gas-solid system, we obtain

$$d\gamma_{SA} = -\Gamma_X d\mu_X \qquad [28]$$

This is the same as the relation obtained by adopting the convention $\Gamma_{H_2O} = 0$. Thus our adoption of this convention means we are defining collector adsorption Γ_X as the total surface excess Γ_X^A experimentally measured less the amount of collector $c_X\,\Gamma_{H_2O}^A$ which may be considered to be associated with the surface excess of adsorbed water in the same proportions as the two components are present in solution. For practical flotation systems, c_X is generally less than 10^{-4} and often is of the order of 10^{-5} or 10^{-6}. Thus the numerical difference between Γ_X and Γ_X^A, the two definitions of quantity adsorbed, is of no practical significance.

Even if no assumption is made as to the order of magnitude of c_X, this analysis still proves that an increase in contact angle on collector addition requires greater adsorption at the solid-gas interface than at the solid-water interface. Comparison of Eq. 5 for the solid-water interface and Eq. 27 for the solid-gas interface shows that an increase in θ and a corresponding decrease in $(\gamma_{SA} - \gamma_{SL})$ require that

$$(\Gamma_X^A - c_X\,\Gamma_{H_2O}^A)\ \text{solid-air} > (\Gamma_X)\ \text{solid-water}$$

or

$$\Gamma_X^A > \Gamma_X + c_X\,\Gamma_{H_2O}^A \qquad [29]$$

In other words, when the adsorption of water is taken into account, our case for a greater adsorption at the gas-solid surface is *strengthened rather than weakened.*

The authors also wish to express their appreciation of the interest shown by Professor Wada in this problem and his contribution is acknowledged gratefully. Professor Wada, however, stresses the kinetic aspect of the problem more than the thermodynamic approach which was followed in our paper.

Manganese Upgrading at Three Kids Mine, Nevada

by S. J. McCarroll

DISCUSSION

J. Bruce Clemmer, J. B. Rosenbaum, and C. H. Schack *(U.S. Bureau of Mines, Salt Lake City)*—We have watched with considerable interest Three Kids development of Manganese Inc. and have been impressed by the ingenuity with which management and staff have solved their many obstinate handling problems.

Mr. McCarroll has been lenient in describing the difficulties faced by the company in developing a process for beneficiating Three Kids ore. The operation, equivalent in size to a 20,000-ton sulphide ore plant, is treating a unique ore by a flotation process pioneered by Manganese Inc. Perhaps the magnitude of the conditioning and flotation problems peculiar to Three Kids ore can be better appraised when it is pointed out that two-thirds to three-fourths of the ore is floated in the rougher step and that the four stages of cleaning which follow are equivalent to floating the whole plant feed two to three times.

Our experience in both laboratory and pilot plant flotation of Three Kids and other manganese ores leads us to agree with Mr. McCarroll that suitable conditioning is a prime requisite for successful oil-emulsion flotation. The difficulties of proper conditioning were demonstrated in pilot plant processing of Artillery Peak manganese ore at our Boulder City, Nev., experiment station. Over three months of experiments and extensive plant revision were required before satisfactory conditioning was obtained in the 25-ton-per-day plant. Such factors as time, type and intensity of mixing, aeration, surface to volume relationship, and power input are still being studied by the Bureau of Mines in hopes of learning how best to condition manganese pulps for oil-emulsion flotation.

Other factors affecting flotation of manganese with oil emulsion also deserve attention. A few of these are the influence of stability and proportion of ingredients in the collector emulsion, the feasibility of grinding

with fuel oil prior to conditioning with an emulsion of tall oil and a wetting agent, the possibilities of stage conditioning with the collector, and the importance of aeration in roughing and cleaning.

The highly friable and porous nature of the Three Kids ore accentuates crushing, grinding, conditioning, flotation, and handling problems. Further degrading of the fine ore apparently occurs even after fine grinding and may account for a part of the difficulty of conditioning and flotation and the large amounts of reagents required.

R. R. Lloyd, Chief, Electrometallurgical Branch, Metallurgical Division, Region III, U.S. Bureau of Mines, has suggested that the difficulty in controlling the temperature and ring formation in the upper end of the nodulizing kiln may be due to the exothermic oxidation of the reduced manganese in the calcined concentrate. If this is so, a remedy would be to permit the calcine to reoxidize prior to charging into the nodulizing kiln. Such a plant change would be difficult but may offer a solution to the continuing problem of ring formation in operating the nodulizer.

We are aware that cursory sintering tests were made by Manganese Inc. prior to selecting the two-stage calcining and nodulizing procedure for lead removal and agglomeration. In view of the still serious difficulties with calcining and nodulizing, it may be pertinent to re-appraise the sintering work. Although the physical strength of sinter is appreciably less than that of nodules, there is reason to believe the sinter strength is adequate. Our small scale tests show that lead can be almost completely removed by addition of about 2 pct salt to the sinter feed. The possibility of volatilizing some manganese along with the lead is acknowledged. The fact that gases from the sinter machine windboxes can be segregated, if desired, may facilitate separation of lead fume from dust. Although major problems undoubtedly would arise in sintering, we confess to being swayed by the prospect of at least having the problem where you can get at it.

S. J. McCarroll *(author's reply)*—The discussion by Clemmer, Rosenbaum, and Schack points out definite lines of investigation which could result in improvements in our process of manganese recovery.

Experimental work on conditioning and grinding is being carried on in both the laboratory and in the mill.

Since the paper was written improvements in operating techniques have resulted in better kiln operation. The idea of exothermic heat resulting from the oxidization of the manganese monoxide, as advanced by Mr. Rex Lloyd, is undoubtedly true and serious thought has been given to the proposal he makes.

With the knowledge we now have of the problems of reagent removal and nodulizing no new installation would be made without thorough investigation of sintering in preference to nodulizing.

A Physical Explanation of the Empirical Laws of Comminution

by D. R. Walker and M. C. Shaw

DISCUSSION

Dimitri Kececioglu *(Allis-Chalmers Manufacturing Co., Milwaukee)*—The idea of applying metal cutting theory to comminution and vice versa is very impressive. Among others, the demarcation of wheel-grinding data into two regions, namely Kick's and Rittinger's, draws attention here. A study of Figs. 6 to 9 shows the following: In Fig. 6, 1 point out of 8; in Fig. 7, 4 points out of 26; in Fig. 8 only 1 point out of 13; and in Fig. 9 practically none out of 13 points fall out of curve No. 2. Also when grinding particles smaller than 5×10^{-4} in. it is difficult to measure very accurately the grinding forces, chip geometry, and particle size. In view of the above it would be difficult to make a positive and distinct demarcation of the wheel-grinding data into two regions, namely, Kick's and Rittinger's.

The idea of using machine grinding to determine the comminution properties of materials is indeed unique. The following cautions come to mind, however, regarding applicability of machine grinding data to comminution:

1—Grinding gives uniform particle size, whereas comminution gives a variety of particle sizes in the same process. The fact that grinding gives uniform particle size is not necessarily an advantage because in comminution the overall energy under actual comminution is desired and comminution invariably results in the production of a variety of particle sizes.

2—In grinding, a small particle is ground off a comparatively large surface. In comminution small bodies are broken down to smaller bodies. This makes it difficult to establish a concept of reduction ratio in machine grinding.

3—The mechanics of machine grinding involve direct shearing action under very high strain and comparatively high compression at the shear zone, whereas comminution involves direct dynamic compression and surface friction with comparatively little strain and shear. Consequently the mechanics of machine grinding appear to be different from those involved in comminution. Hence one might arrive at erroneous conclusions when comminution characteristics are evaluated from machine grinding results.

It is noticed also that functions other than Eq. 16 which do not involve the concept of direct proportionality of useful energy input to new surface area created (Eq. 2 and particularly Eq. 12) may be derived to fit the data of curve No. 2. One such relationship may be $u = ct^{-n}$, where c is a positive constant and n is a positive constant less than unity. This, of course, is made possible because data of a very small range of particle size are being fitted by the curves of No. 2 and the hereby suggested relationship. A sound comminution theory should be applicable to a relatively large range of particle size.

George B. Clark *(Department of Mining Engineering, University of Illinois)*—While the experimental work of Walker and Shaw in this article is of intrinsic value in revealing certain fundamental aspects of machine grinding, milling, and related processes and in adding to the store of information concerning the comminution of solid materials by this particular process, it appears doubtful that the results of their research can be employed to verify either Kick's or Rittinger's theories. Both these laws were originally developed and proposed on the basis of very simple, idealized hypotheses. The creation of parallel idealized physical laboratory conditions which would be required for either their proof or disproof appear to be well beyond the scope of human ingenuity, at least at present.

One great weakness of the imputed verification is that the spread of the points plotted on the graphs is not sufficient to permit accurate curve fitting. The range of particle sizes included in the belt and wheel grinding is not extensive enough to show any general trends, particularly for the horizontal portion of the curves, said to represent Kick's laws.

That the milling, grinding, and turning data obtained by Walker and Shaw for both metals and minerals happen to comply with certain interpretations of these two theories under certain specialized conditions

of the comminutive process is merely fortuitous. The basic factors which govern the energy consumption are concerned with different physical properties than those recognized by the two theories in question; hence their applicability cannot be supported. There are other objections, as well, to the employment of these data to support the much-debated *laws* of crushing. The obvious incompatibility of these results and the development of Kick's theory is that it was devised on the assumption that all strains which contribute to failure in the comminuted material are within the elastic range. Thus the occurrence of plastic flow in the tests in question, which the authors use as a basic reason for drawing a close parallelism between machine grinding and other comminution processes, is one more specific reason why their data cannot be used to verify this crushing theory.

The statement of Walker and Shaw that the machine grinding process offers a precise means for evaluating the grinding characteristics of various materials is also open to question. Outwater and Shaw[8] show that the chip and surface temperatures in machine grinding of metals are very high, high enough to alter the physical properties of the metal. This factor alone would appear to make the applicability of machine grinding as a grindability test a very limited one, that is, restricted to those grinding processes where temperature and grinding speed conditions exist which are similar to those in the actual grinding test. To attempt to apply them to dissimilar tests would be extrapolating through a maze of unknowns to a region of processes at a great distance. Such extrapolation is not justified by the arguments or data presented.

It is also possible that of the energy absorbed by these grinding processes the amount involved in heat losses is so great that the actual energy used in comminution is small enough in comparison to be negligible. These grinding processes, therefore, may be simply measures of deformation rather than comminution, the factors involved in creating particles of "like technological state" being obscured by the energy losses of friction, deformation and other processes, which are not dominating factors in ordinary crushing and grinding.

Another factor which leaves these apparent verifications of the crushing laws open to criticism is that the comminution in each case was accomplished in one step. Such a process ignores the effect of the relative size of the particle being chipped and the body from which it is being separated. In other words the strain both in magnitude and distribution depends upon the geometrical configuration of the bodies being separated as well as the manner in which the forces are applied. One-step crushing or grinding of ores or other solids is an impossibility, except perhaps in the case of explosive action under special conditions. In this case the material might be strained and broken in a single-step process, but even so, the effects of plastic flow behavior of solids under high pressures would come into play. The laws of probability would enter in with respect to particle size distribution, distribution of imperfections in the material, variation of strain distribution from the theoretical, and so forth.

Bond[9] has shown that the mathematical expression employed by Walker and Shaw for Rittinger's theory is in error and that the corresponding curve should be concave downward on a log-log plot. It is asymptotic to a line with a slope of —1 as the product size becomes small relative to the feed size.

For all *stepwise* comminution processes the graphical representation of Kick's theory is also a curve which is concave downward. It could not be a horizontal line because the mathematical expressions proposed by this theory represent the energy of comminution as being a function of the reduction ratio as well as the volume. Hence for the results of Walker and Shaw to be analogous to other comminution processes, the points plotted as being representative of Kick's theory should be shown as points on a family of curves which are concave downward.

To summarize, therefore:

1—The wheel and belt comminution data cannot be used with accuracy to verify Kick's and Rittinger's laws because the physical state of the material is different from that postulated in the development of the laws.

2—The spread of the data is insufficient to justify curve fitting or application of mathematical expressions of these laws.

3—The mathematical and graphical interpretation of Rittinger's comminution theory is in error, and the graphical representation of Kick's theory as a horizontal line is likewise not in agreement with applied interpretations of his hypothesis.

4—The very specialized test conditions involved in belt and wheel grinding would seem to obviate these as standardized grindabiliity tests.

5—None of the laws of comminution have been proved, but they should serve as valuable bases for further theoretical and laboratory investigation.

[8] M. C. Shaw and J. O. Outwater: Surface Temperatures in Grinding. *Trans.* ASME (1952) 74, p. 73.
[9] F. C. Bond: Personal communication.

Fred C. Bond *(Allis-Chalmers Manufacturing Co., Milwaukee)*—The authors have translated considerable study and research in machine grinding operations, principally of metal, into the field of mineral comminution. In doing this they have assumed without any published proof that the depth of cut of a machine tool grinding minerals equals the particle diameter of the ground product, irrespective of the tool speed, grinding surface, and other operating conditions. As would be expected under this assumption, they find large discrepancies between the energy required by a fine grinding wheel and coarser belt-grinding tool.

According to the Rittinger theory the total specific energy varies inversely as the product diameter. Eq. 12 is not a statement of the Rittinger theory because of the term $-u_o$. This term causes the upward curvature of the plotted lines in Figs. 6, 7, 8, and 9. If u represents the total specific energy in reducing from an infinite feed size to product size t the term u_o should vanish, and the Rittinger plotted line will be straight with a slope of 1:1. If u represents the specific energy increment in reducing from a finite feed size, the term u_o corresponds to the energy in the feed, and should be added to u, resulting in a curve which is concave downward and which does not fit their experimental data. The error seems to have arisen from their assumption of a positive constant of integration in Eq. 10. The term $V_2 - V_1$ in Eq. 8 equals zero from Eq. 7.

When starting with a finite area A_1, volume V_1, and particle size t_1, Eq. 8 should be

$$u = K_2\left(\frac{A_2}{V_2} - \frac{A_1}{V_1}\right)$$

and from Eq. 11

$$u = \frac{K_2}{t_2} - \frac{K_1}{t_1}$$

and

$$u + \frac{K_1}{t_1} = \frac{K_2}{t_2}$$

or

$$u + U_o = \frac{K_2}{t} .$$

Without evidence regarding the actual particle sizes produced, and with the error in the Rittinger statement, this commentator is unable to accept their conclusions regarding the relationship between the energy input and the three comminution theory criteria of 1—surface area produced, 2—particle volume broken, and 3—crack length formed.

D. R. Walker and **M. C. Shaw** *(authors' reply)*—The authors wish to thank the several discussers for their thought-provoking comments. In answer to Mr.

Kececioglu's observations, we can see no reason why it is more difficult to measure forces and calculate mean undeformed particle shape in fine grinding than in coarse grinding. Although it is true that a range of particle sizes results from comminution, while machine grinding produces particles of essentially constant size, the comminution energy for a range of sizes may be synthesized from the individual values of energy requirements as determined in machine grinding. In comminution there is an appreciable expenditure of deformation energy and friction energy as in machine grinding, so in this sense the two processes are similar.

As stated in the paper, the authors are in agreement with Professor Clark's observation that neither Kick's nor Rittinger's rule has been physically verified, since the original models upon which these rules were formulated are in error. As the title of the paper states, this work is concerned with a physical explanation of the real reasons why the so-called laws of comminution happen to fit comminution experiments. While it is true that extremely high surface temperatures of extremely short duration are encountered in machine grinding, it is equally true that transient high surface temperatures will be experienced in comminution. Professor Clark's statements regarding the difference in physical state because of the plastic deformation of machine-ground particles apparently stems from the common belief that comminution is an essentially brittle process. It has been one of the objectives of this paper to point out that plastic deformation is to be expected in nominally brittle materials when the particle size is small.

Another objective of this paper was to call attention to the physically important variable of particle size rather than surface area. It is thus gratifying to note that Mr. Bond has stated Rittinger's rule in terms of product diameter. Except for a typographical error corrected by Mr. Bond, Eq. 8 is an expression of Rittinger's rule. Mr. Bond is correct in pointing out that the constant u_o in Eq. 12 should be negative if this equation corresponds exactly to Rittinger's rule. Since u_o is found to be positive when Eq. 12 is fitted to experimental data, it follows that Eq. 12 does not represent Rittinger's rule, but rather represents a hyperbola which differs by a constant from the hyperbola predicted by Rittinger's rule. This result can only mean that Rittinger's rule fails to agree with experimental data by the amount of the constant. That Rittinger's rule cannot be strictly true can be further observed from Mr. Bond's last equation. If the value of t were made very large, $K_{s/t}$ would approach zero and u would have a negative value. The explanation for this anomaly lies in the fact that Rittinger's rule fails to agree with experimental data by a constant. Since most applications of Rittinger's rule are for energy differences, the offending constant disappears when the difference is taken.

F — Coal

Underground Electrocarbonization of Coal and Related Hydrocarbons

by T. C. Cheasley, J. P. Forrester, and Erich Sarapuu

DISCUSSION

John G. Tripp—The concept of converting fuel deposits *in-situ* to useful forms of energy and chemical raw materials has important possibilities as a means of stretching out our fuel resources. The authors have worked in this field for years and in fact are among the pioneers in developing *in-situ* techniques based on underground electrocarbonization. Their efforts have aroused great interest in this subject and their contributions will undoubtedly gain even more significance as time goes on.

The 4 to 5 billion tons of petroleum are proved reserves, that is, they are completely defined and could be produced economically with present methods of recovery. Reserves of coal, tar sands, and oil shale shown in Table I are not equivalent to *proved reserves* as the term is used in the oil industry. Incidentally, the 4.3 billion tons of oil shown in the tabulation is equal to about 28 billion barrels of oil and, therefore, presumably does not include reserves of natural gas liquids estimated at about 5 billion barrels. The relative comparisons would be more complete if these liquids together with the reserves of natural gas were included in the tabulation.

It is stated in the report that production costs of the order of 10¢ per million Btu's could be expected from a large industrial electrocarbonization installation. The authors state that the operation probably could be operated successfully and at a profit.

I would like to express my view that for the operation to be profitable extremely favorable conditions would have to prevail. For example, the coal seam would have to be thick and close to the surface in order to minimize drilling costs; the deposit should be dry and so situated that there would be little, if any, inflow of ground water; and the seam would have to be relatively gas tight to avoid excessive gas leakage. Notwithstanding the tremendous strides that have already been made, it is my hope that the authors plan to continue their development work. Experiments on underground electrolinking and initial carbonization at Gorgas, Ala., indicate a need for improved control over the operation. Subsequent examination of the underground system at Gorgas has shown that the gasification zone did not always follow a direct path between the electrodes. Undoubtedly the control of the in-situ thermal processes can be improved by further development work.

With respect to the electrocarbonization of oil shale it may be well to point out that the cost of mining oil shale is estimated to be 50¢ to 60¢ per ton. These are low mining costs as compared to coal. Thus it would appear that there is less economic incentive for applying in-situ thermal methods to oil shale than there is for applying these methods to coal. As is the case with in-situ methods for petroleum, the product from the shale having the most economic interest would be liquid oil rather than gas.

The authors point out that for distances up to several hundred miles it is cheaper to transmit energy as electricity than it is to ship energy as coal. This is a very important point, and with coal shipping rates rising periodically, it could prove increasingly important in the future.

I believe that the authors deserve great credit for their outstanding work on underground electrocarbonization. The knowledge resulting from their efforts is of great value to our fuels technology.

James L. Elder—The experimental work conducted by the authors on the electrolinking-carbonization of carbonaceous materials in situ is of great interest. The method is of great importance in the development

of processes for the underground gasification of coal and probably will have application in the treatment of oil shale and oil-bearing sands. However, this discussion is limited to applications of the method towards processes for underground gasification of coal. The procedure is an excellent one for preparing an underground unit for operation. The authors should be congratulated for their experimental development work.

In the paper an effort has been made to develop phrases which adequately describe the phases of the electrical procedure. The term *electrolinking* is used to describe the formation of channels or linkages within the carbonaceous strata in situ. This term is a good one and I believe adequately describes the phenomenon which takes place upon the initial application of an electric current to a carbonaceous deposit. The term *electrocarbonizing* has been used to define the second phase of the process where, following the development of a good electrical path, extensive electrical heating is used to drive off the volatile components of the carbonaceous matter. It also adequately describes the phenomena taking place. A third term, *underground electrocarbonization*, is then used in an inclusive manner to cover electrolinking, electrocarbonizing, electrogasifying, and gasification operations conducted with air, oxygen, or steam without the accompanying use of an electric current. These latter operations cover the true gasification functions whereby a solid fixed carbon is chemically changed to a gas. Application of an electric current alone does not result in gasification of fixed carbon and it can be used only in a secondary manner, to supply heat for the reactions involved. Use of the term *underground electrocarbonization* in an inclusive manner is confusing; it should be applied only where the phenomena involved are completely dependent on action of electric current.

It is stated correctly that it is reasonable to expect gas production costs of the order of 10¢ per million Btu for a commercial plant comprising 30 to 50 coal underground gasification units. This figure is based upon the underground gasification of coal with air following initial electrolinking-carbonization as a means of site preparation. It is expected that in such an operation approximately 1 pct of the net energy produced would occur during the electrolinking-carbonizing phase and 99 pct of the net energy would be from the phase involving gasification with air. In underground gasification of coal the electrolinking-carbonization is by far the best method yet developed for site preparation; it is through use of this method that the most successful gasification operations have been conducted. The quantity production of useful gaseous products, however, appears dependent upon subsequent gasification operations rather than on preliminary site preparation phases.

It has been stated that at the present time there is insufficient experimental data to permit the comparison between uniflow, reversal of flow, or recycling operations. Recycling of product gas during underground gasification might improve the percentage conversion of carbon to carbon monoxide and to date no experimental trials have been conducted that I know of. The improvement would have to be balanced against the costs of recycling which will include repressuring product gas and probably cyclic operations using several unit systems simultaneously. The nearest approach that has been tried experimentally is a cyclic operation involving air blow and steam runs, data for which is given in Table IV. Consideration of the steam run data can be made to estimate the value of recycling product gas, for the operation would have to be carried out in much the same manner. Reversal of flow operations has been tried experimentally on numerous occasions and generally has resulted in a lower quality product. Reversal of flow is usually recommended as a method of decreasing heat losses, but in the experimental trials I believe the opposite results have been achieved.

Continued efforts are being made at Gorgas to determine the location, size, and shape of reaction zones resulting from underground gasification. As stated, tests were made using a modified zero potential electrical method. This has been followed by an extensive drilling program which is still underway. The results of the drilling program do not agree with the electric method. Reference is made to Fig. 3, right; test drilling shows that coal consumption near the input borehole is a maximum and the width decreases as we move downstream towards the outlet. The shape illustrated should show a larger area of coal consumed near the inlet. Using the inlet borehole as a pivot point, the direction of the path which was actually developed is displaced approximately 22° to the right of that shown. The discharge point for the system is probably an old system existing beyond the exhaust borehole indicated and this exhaust borehole is probably not directly connected to the system. The location of the exact discharge point is not yet definitely known. With respect to Fig. 3, left, the reaction path followed has been found rotated exactly 90° to the left, the input borehole being used as a pivot point. In this case the exhaust borehole shown does not appear to be directly connected to the system. The discharge point for the system represented in Fig. 3, left, is actually at the input borehole shown in Fig. 3, right. The test drilling work again shows a maximum coal consumption adjacent to the input borehole.

The changes in path location as well as the existence of carbonized areas influences the resistance-time curves 3, 4, 5 given in Fig. 1. The effective lengths of these paths are no longer the 150 ft shown. For curve No. 3, the length of path must be less than 150 ft and possibly about 120 ft, although it is not now possible to determine this length with any accuracy. For curve No. 4, the length remains 150 ft, for no change in path location has been found. For curve No. 5, the effective length of path must be greater than 150 ft and may approach 175 to 190 ft. It should yet be possible to determine this path length within moderate limits and work is continuing with this objective in mind. It is very likely that the differences in length of path greatly influence the pre-linking time shown in Fig. 1. The history of the electrolinking tests at Gorgas indicates that curve No. 3 of Fig. 1 was probably subjected to more influence by previously carbonized areas than the tests represented by curves 4 and 5.

Longwall Mining and Mechanization with Special Reference To Nova Scotia

by Frank Doxey

DISCUSSION

Louis A. Turnbull *(U.S. Bureau of Mines, Washington, D. C.)*—This paper is a well-organized concise discussion of progress in mechanization at coal mines in Nova Scotia during the last 50 years. It shows the perseverance and resourcefulness of those responsible for the production of high-quality coal essential to steel production in the area. The development of the Dosco miner and its use in longwall mining at the Sydney

coal mines is of particular interest. Some of the operating data given deserve attention. Average productivity of machines and workers is not discussed in detail except that the statement is made that maximum production per shift with the Dosco miner has been 587 tons and maximum weekly production 2860 tons (5 days), an average of 572 tons per shift. If the crew of 55 men on a mechanized unit mentioned near the end of the report is compared with these production figures, the production for the best operating period would be 10.4 tons per man per shift at the face including packing, roof control, and moving conveyors. The statement also is made that there are 96 men in the hand loading crews and 55 men in the mechanical loading crews. Mining with the Dosco miner has reduced the number of men required 43 pct, and if the production for hand loading units vs mechanical loading units is about the same the reduction in labor charges has been substantial.

It is difficult to compare mining operations in Nova Scotia with those in the United States because physical conditions are not at all similar. However, several operators in the United States have experimented with modified longwall systems recently, using European type of equipment. The Meco-Moore machine was used experimentally in West Virginia, the Samson stripper in central Pennsylvania, and the German coal planer is in operation at two mines in West Virginia and one in central Pennsylvania. Data on the operation of the planer in the Pocahontas No. 4 bed in West Virginia are available.[1] A panel 328 x 1455 ft was mined with this device. The coal bed averages 34 in. thick and is flatlying. Maximum production for one shift was 842 tons and average for the entire panel 582 tons per shift for 93 shifts. Average production per manshift was 18.0 tons during planer operation. Including the time spent in developing the panel overall production was 13.0 tons per man per shift which was about 20 pct higher than the average manshift production obtained in conventional mining at this mine. Yielding metal props with fitted steel cap members and wood cribs were used to support the roof. The props were moved forward as the face advanced and a straight break line was almost always maintained. Roof action was good because the roof rock broke and fell almost as soon as the jacks farthest from the face were removed and replaced along the conveyor line. A German-made double-chain panzer conveyor used at the face was at least partly responsible for the good operating results obtained. This conveyor was moved automatically toward the face by pneumatic jack.

Mr. Doxey mentions in the final pages of his paper that improvements in face and intermediate transportation of coal away from the Dosco miner are being studied and will be made. To this extent at least some of our problems are similar. Development of high-capacity loading and mining devices has introduced the problem of moving the coal from the machine to the main transportation system to the end that the loading or mining machine may be kept in operation as continuously as possible throughout the shift.

A number of coal mine operators have expressed interest in modified longwall mining as a means of improving the percentage of recovery of coal and at the same time effecting improvements in productivity per man and machine. It appears that the Dosco miner or some similar device might be suitable for use in the United States where physical conditions of the coal bed and surrounding strata are favorable. At any rate this paper and others presented to the joint Canadian-United States meeting on coal mining undoubtedly will benefit both groups participating in these discussions.

[1] Wilber A. Haley, James J. Dowd, and Louis A. Turnbull: Modified Longwall Mining with a German Coal Planer (Flow) in the Pocahontas No. 4 Coal Bed, Helen, West Virginia. U. S. Bur. Mines. *R. I.* 4922 (1952) 13 pp.

Cleaning Various Coals in a Drum-Type Dense-Medium Pilot Plant

by M. R. Geer, W. A. Olds, and H. F. Yancey

DISCUSSION

John Griffen *(Consulting Engineer, Pittsburgh)*—This paper presents much data offering tentative answers to many technical questions that have been raised by the employment of dense-medium processes and, particularly, by use of the drum-type vessel. It is to be hoped that the Bureau of Mines will publish complete detailed data for the 50 or more tests so that it may be studied and analyzed by everyone interested.

Such study will take considerable time and effort. The writer is therefore obliged to confine his discussion to a few points he has noted as being open to question. The data in Table VI which shows the recovery efficiency for the 1 to ½-in. fraction as 100 pct raised a question concerning the accuracy of these calculated figures; washability data on the washed coal and refuse show that such a figure is impossible. When the ash content of the misplaced material in the washed coal is higher than that of the misplaced material in the refuse, recovery efficiency must be less than 100 pct. A study of the data in Tables III, IV, V, and VI suggest that this error probably arose from slight errors in calculating the data in Table V and more probably from the small scale used in plotting the washability curves from the Table V data.

Using the size consist data and the weight and ash percentages of individual gravity fractions of each size as given in Tables III and IV and the reported recovery of 80 pct of washed coal, a revised Table V was calculated, carrying weight and ash percentages to hundredths of a percent instead of rounding off at tenths. This washability data for each size of Bellingham reconstituted feed was then plotted on large scale washability curves.

From this data the new Table XIV was prepared, which agrees very closely with data shown in Table VI except for two important values, theoretical yields and recovery efficiencies. As will be noted the recovery efficiencies in Table XIV are significantly lower than those given in Table VI.

A study was then made of the data showing distribution of the specific gravity fractions to washed coal and refuse. This study indicated that when this distribution was based on the fractions as tested at 0.1-sp gr intervals the number of points obtained were insufficient to establish the distribution curves with reasonable precision, which in turn would affect the accuracy of the values for the specific gravity of separation and the error areas.

The data in Tables III and IV of the paper and curves prepared therefrom make it possible for the specific gravity fractions to be subdivided further. This was done using 0.05-sp gr fractions and the distribution data so obtained is given in Table XIV, with resulting values for the specific gravity of separation and the error areas.

The writer believes that the derived values shown on Table XIV are more accurate than the correspond-

ing values given in Table VI, but the significant point is that from the same original data derived values are obtained that are different. These differences are such as to indicate lower recovery efficiencies and less sharp separations. This points to the fact that the criteria now being used to evaluate washing performance are dependent upon graphical representation of observed data and that unless great care is used in constructing the graphs, significant errors can arise.

The writer believes this is particularly true in evaluating the performance of heavy medium processes with their much more precise separations. To reduce the chances for errors due to graphing, the float and sink fractions should be determined, certainly close to the probable gravity of separation, at intervals less than 0.1 sp gr, certainly not greater than 0.05 and preferably at 0.02 sp gr. It is believed that tests conducted in this manner, coupled with graphing of data on scales large enough to reduce errors to a negligible value, will render possible derivation of the criteria of washing performance with an accuracy that will give these values practical significance.

The writer has no doubt that most of the conclusions drawn by the authors are valid. However, if the derived values of the other tests are subject to variation in value such as shown by those in Tables VI and XIV, some conclusions may be in error or at least not supported by the original data.

The tentative conclusion that viscosity has no significant effect on washing results does not seem to be supported by the evidence submitted in the paper. The tests 43 and 44 summarized in Table XIII and compared for this purpose both show high error areas. The bath medium of test 44 is supposed to be of low viscosity as it was made up of clean medium, but the specific gravity differential in the bath was only 0.01. Such a low specific gravity differential can be obtained only by grinding the magnetite extremely fine, which entails a high viscosity of bath medium, or by agitating the medium strongly, which impairs the sharpness of separation, or by both. No viscosity figure nor explanation for this low differential is given.

Test 41, which was made on the same coal with a gravity differential of 0.05 (see Tables VII and X) gave much better results. It is also unfortunate that Bellingham coal, which had shown surprisingly high degradation sufficient to affect markedly the error areas, was selected for the comparison on the effect of viscosity or contamination of the medium.

It would appear that a conclusion could have been justified only if the comparison were made on a coal which had no significant influence due to degradation, if the viscosities of the two bath mediums had been determined, and if all other operating variables, including drum speed or agitation, were equal.

The writer thinks it is significant that the gravity differential in the bath seems so important, particularly when separating between 1.55 and 1.60 sp gr. It should be pointed out that no tests are reported for the range from 1.41 to 1.50 nor above 1.60 sp gr. It is therefore possible that a definite gravity differential within the bath has a beneficial effect on performance over a wider range of separating gravities than the authors suggest. However, in view of the error-area value of 15 for test 41 shown in Tables VII and VI and the error area value of 20.4 shown in Table XIV, the error areas shown in Table X may be open to question. Should further checking of the data of these tests or additional tests confirm the value of a gravity differential in the bath, it would be a confirmation of the views of K. F. Tromp of Holland, inventor of the Tromp heavy medium process, that the best washing results are obtained in a bath medium with a differential density. A differential in density is obtained by keeping the bath medium both fluid and relatively quiescent. A fluid bath must have a low viscosity. A further requirement for best washing results is that the bath be kept free from congestion with intermediate density materials.

Table XIV. Summary of Performance of Pilot Plant on Bellingham Coal

Screen Analysis	Size Fraction, In. 2 to 1	1 to ½	½ to ¼	¼ to 0	Composite 2 to 0
Feed, pct	22.42	39.90	24.70	12.98	100.00
Washed coal, pct	23.30	41.80	24.30	10.60	100.00
Refuse, pct	18.90	32.30	26.30	22.50	100.00
Ash Content					
Feed, pct	20.47	20.09	24.33	33.11	22.91
Washed coal, pct	15.40	14.20	15.83	13.46	14.80
Refuse, pct	45.43	50.61	55.75	70.14	55.37
Yield of washed coal, pct	83.14	83.81	78.70	65.33	80.00
Theoretical yield, pct*	84.00	84.40	79.40	66.00	80.69
Recovery efficiency, pct	98.98	99.30	99.12	98.98	99.14
Distribution Data, Pct Recovered in Washed Coal					
Under 130		100.00	100.00	100.00	100.00
1.30 to 1.35	100.00	100.00	100.00	100.00	100.00
1.35 to 1.40	100.00	100.00	99.90	99.47	99.92
1.40 to 1.45	100.00	100.00	99.87	99.55	99.93
1.45 to 1.50	97.55	98.21	99.55	97.84	98.42
1.50 to 1.55	42.01	47.68	80.04	80.72	57.32
1.55 to 1.60	4.98	25.36	51.92	86.04	31.91
1.60 to 1.65	0.0	0.0	10.89	17.31	5.16
1.65 to 1.70	0.0	0.0	0.0	12.88	1.77
1.70 to 1.75	0.0	0.0	0.0	3.78	0.57
1.75 to 1.85	0.0	0.0	0.0	4.72	0.73
1.80 to 1.85	0.0	0.0	0.0	10.54	1.79
Over 1.85	0.0	0.0	0.0	0.0	0.0
Specific gravity of separation	1.518	1.520	1.577	1.584	1.537
Error area	10.0	18.8	20.3		20.4
Sink in washed coal feed, pct**	1.41	1.67	1.18	1.31	1.52
Float in refuse, feed, pct**	1.53	1.04	2.00	1.56	0.96
Total misplaced material, feed, pct**	2.94	2.71	3.18	2.87	2.50

* Float-and-sink yield at ash content of washed coal.
** At specific gravity of separation.

The writer regrets that more time is not available for study and discussion of this paper. He urges that the authors arrange for publication of the complete data of all the tests so they can be widely studied and evaluated. The authors are to be commended and congratulated on the ground covered by their paper which the writer hopes will stimulate others to present similar studies of the performance of commercial heavy medium plants.

M. R. Geer, W. A. Olds, and **H. F. Yancey** *(authors' reply)*—The figures in Mr. Griffen's Table XIV differ somewhat from those in the original table VI because he carried his calculations out to the second decimal place and interpolated on the yield-specific gravity curves to simulate the use of additional baths in the float-and-sink tests. These techniques raise the question of whether our chain of data can be stronger than its weakest link: the accuracy with which the yield of washed coal is known. All the performance criteria depend on the yield of washed coal, and generally this figure cannot be determined closer than about 1 pct. Carrying out calculations of yield and efficiency to four significant places simply is not justified by the precision of the basic data.

The practice of interpolating on yield-specific gravity curves is hazardous, as the following illustration demonstrates. In one washing test the usual 0.1 intervals of specific gravity were used for the float-and-sink tests, supplemented by baths of 1.55 and 1.65 sp gr; the distribution data were calculated, using these two additional densities. Then, following the system employed by Mr. Griffen, the yield-specific gravity curves for the washed coal and refuse were plotted, using only the data at each 0.1 interval and ignoring the available data at 1.55 and 1.65. Interpolated values for 1.55 and 1.65 sp gr were read from these curves and the distribution data calculated for comparison with those actually determined in testing. The fol-

lowing tabulation shows that the interpolated values were far from accurate. Errors in distribution amount-

Specific Gravity	Percent to Washed Coal	
	Determined	Interpolated
Under 1.30	99.7	99.7
1.30 to 1.40	99.8	99.8
1.40 to 1.50	99.7	99.7
1.50 to 1.55	97.4	91.3
1.55 to 1.60	59.6	68.1
1.60 to 1.65	29.5	38.7
1.65 to 1.70	17.4	10.0
1.70 to 1.80	6.7	6.7
Over 1.80	0.3	0.3

ing to 9 pct occurred through the interpolation, because of improper assumptions regarding the shape of the yield-specific gravity curves. The likelihood that Mr. Griffen's interpolations were similarly inaccurate is demonstrated by the irregularities in distribution data obtained. For example, his distribution data for the ¼-in. to 0 size shows 10.54 pct of the 1.80 to 1.85 fraction reporting to washed coal, in comparison with a recovery of only 4.72 pct of the next lighter fraction. A similar inconsistency occurs in the composite of all sizes. Such irregularities simply are contrary to the known performance characteristics of washing equipment and point unmistakably to errors—in this case false assumptions about the shape of the yield-specific gravity curves on which the interpolations were made. Thus the authors do not concur in Mr. Griffen's belief that the data in his table are more accurate than the original.

Our observations on the influence of the viscosity of medium were offered as being only tentative, and we agree entirely with Mr. Griffen that the Bellingham coal was not an ideal one on which to judge the influence of viscosity; it simply was the only coal on which the data were available. Additional work now under way will provide a great deal more information on the role of medium properties.

Frontiers in Heat Extraction from the Combustion Gases of Coal

by Elmer R. Kaiser

DISCUSSION

G. A. Vissac *(Consulting Engineer, Vancouver, B. C.)* —Some of the data presented in this paper, in connection with cost studies of washed coals, should be clarified and qualified.

Washing a raw coal down to a low ash content may mean higher losses in refuse, but the cost of the operation may be more than compensated by the increased value of the resultant product. However, careful engineering and accurate machinery are required to insure maximum profits. Washing at a low gravity separation demands a very accurate automatic process, chiefly when the small sizes of coal are being handled.

Coal consumers today are buying mostly on a Btu basis, cognizant of the fact that low ash contents result in lower installation costs and in savings in handling and transportation charges. The attached Table X gives the actual balance sheets of the various cleaning operations corresponding to various ash contents for the two coals presented in the paper, using the old technique of a two-products separation, or the new technique, now commonly used in Europe, of a three-way separation made possible by new methods and new equipment. Savings in capital costs, handling, and transportation have been calculated on the basis of a delivered cost f.o.b. plant of $10.00 per ton of coal at 16.5 pct ash (actual 1953 average price reported by Consolidated Edison, N. Y.), and of $3.50 at the mine also for 16.5 pct ash coal. In both cases this corresponds to values of 12¢ and 4.2¢ per 1 pct of combustible and per ton, respectively, on delivered and on at the mine prices.

As an example, the middle western coal at 16.5 pct ash, raw, washed down to 4 pct ash in a two-way process will give 53.75 pct of clean coal at 4 pct ash and 46.25 pct of refuse at 31 pct ash. The value of the losses will be $0.4625 \times (100-31) \times 0.042 = \$1.34$. Add the cost of washing, capital, and operation, which is $0.35. The total cost will then be $1.69.

But with a three-ways separation we will have 53.75 pct of first-grade coal at 4 pct ash, 37.25 pct second-

Table X. Cost Studies of Raw and Washed Coals at Two Mines. Balance Sheet

Clean Coal		Washing Cost per Ton of Clean Coal	Cost per Ton of Raw Coal, Old Technique			Cost per Ton of Raw Coal, New Technique		
Ash, Pct	Recovery, Pct of Raw Coal		Credits	Debits	Profit or Loss	Debits	Profit or Loss	
Midwestern Bituminous Coal								
16.5	100	3.50	raw		0		0	
11	91	4.20	0.60	0.455	+0.145			
8.5	86	4.42	0.825	0.535	+0.29			
7.5	83	4.58	0.90	0.632	+0.268			
6.5	79.7	4.75	0.96	0.715	+0.245			
6	78.8	4.80	0.99	0.748	+0.242			
5.5	77.05	4.90	1.02	0.801	+0.22			
5	72.25	5.20	1.00	0.95	+0.05			
4.5	65.55	5.70	0.944	1.22	−0.276	0.60	+0.40	
4	53.75	6.90	0.80	1.69	−0.89	0.53	+0.27	
3.5	43	8.50	0.67	2.13	−1.46	0.52	+0.15	
Eastern Bituminous Coal								
15.5	100	3.55	raw		0		0	
11.0	93	4.12	0.504	0.424	+0.08			
8.5	86.4	4.41	0.675	0.578	+0.107			
7.5	79.7	4.75	0.765	0.79	−0.025	0.423	+0.342	
6.5	71.4	5.26	0.77	1.09	−0.32	0.423	+0.347	
6	68	5.50	0.775	1.21	−0.435	0.423	+0.352	
5.5	61.4	6.06	0.74	1.37	−0.63	0.423	+0.317	
5	54.5	6.78	0.69	1.64	−0.95	0.423	+0.267	
4.5	44	8.33	0.58	2.10	−1.52	0.423	+0.157	
4	33.5	10.81	0.48	2.50	−2.02	0.423	+0.057	
3.5	25	14.55	0.36	2.80	−2.44	0.423		−0.063

grade coal at 21.7 pct ash, and 9.0 pct final refuse at 72.2 pct ash. The value of the losses will be: 0.09 x (100 — 72.2) x 0.042 = 0.105. Add the cost of washing, which is 0.35, and a penalty on the second-grade coal: 0.3725 x (21.7 — 16.5) x 0.042 = 0.075. The total cost will be 0.53.

The cost of the clean coal, on a clean coal basis, as given in the paper, is of no practical value. It was apparently calculated as follows:

$$3.50/0.5375 + 0.35 = 6.90$$

Credits for savings in handling, transportation, and plant cost are, in both cases,

$$0.5375 \times (16.5 - 4) \times 0.12 = \$0.80$$

Accordingly coal washed down to 4 pct ash may cost $6.90 per ton of clean coal, but it will bring a profit of $0.80 — $0.46 = $0.34 per ton of raw coal with a three-ways separation as against a loss of $1.69 — $0.80 = $0.89 per ton of raw coal with a two-ways separation, as against breaking even, if the coal were sold unwashed, if the market would accept it as such.

Rewashing for a three-ways separation is only worthwhile and practical if and when the primary refuse is under about 55 pct ash.

Removing surface moisture instead of ashes is still the best-paying job, as a 1 pct loss in weight corresponds to a 1 pct improvement in combustible.

In all cases, if a certain amount of foreign materials or of moisture is deemed desirable, it can always be added at the consumers plant, and this will result in savings in handling and transportation charges.

As a conclusion, and in terms of global economy, modern methods of washing down to a low ash content will increase profits and widen markets. When dealing with steam power plants, two different conditions must be considered: using a low ash coal in the remote locations, and a second-grade higher ash coal in the vicinity of the coal preparation plant. This is now the common practice in Europe.

Exploration of the Oaxaca Coal Fields in Southern Mexico

by Luis Toron and Salvatore Cortes-Obregon

DISCUSSION

John D. Price *(Colorado Fuel and Iron Corp., Pueblo, Colo.)*—The paper on the coal fields of the Oaxaca district as prepared by engineers Toron and Cortes-Obregon of the staff of the Bank of Mexico bears witness to the thorough and careful way in which the men associated with this organization perform their work. There is little to be added to their paper in way of discussion other than to confirm and amplify some of their statements.

Since the only extensive and well-developed field of coking coal lies in the northeastern section of the country adjacent to Sabinas in the state of Coahuila, it follows that blast furnace plants would be located in that same region. Two such plants are now operating at Monterrey and Monclova, using coke produced at the Sabinas district mines. But the nearer of these two plants is 600 miles from Mexico City and even farther from the center of population. Transportation of products from these mills to the market area is therefore expensive, both because of the distance and the difficulty of the terrain over which it must be carried. The development of an integrated steel industry closer to the center of population has therefore long been a goal toward which the Mexican technicians have been striving.

While the presence of coal of some grade has been reported in many of the states, and many ideas have been advanced regarding its possible uses in iron and steel production, deposits of anthracite in Sonora and the various coals of the Oaxaca district as reported on in this paper are the only ones that have been explored in a serious manner. The coking coal from the Mixtepec zone appears to offer promise of producing a coke which could be used in a standard blast furnace. Several problems are indicated, however:

1—The ash in the coal is high as mined, but indications are that it can be washed to an ash content of 15 pct with a recovery of 70 pct of washed coal.

2—Such washing would increase the volatile content from 17.4 pct to about 20 pct, and in a byproduct oven this should give a coke yield of close to 80 pct with an ash content of coke under 20 pct.

3—A free swelling index of 5 appears low for a good coking coal, and below that of the coals from the Sabinas district, which show between 6 and 9. But washing of the coal should result in an improvement in this regard; in the United States coals from Utah with an index even lower than 5 have made a usable coke.

4—A coal with volatile as low as 17.4 in raw coal and 20 in washed coal would come close to being classed as a low-volatile rather than medium-volatile coal, and low-volatile coals are notorious for their high expansion properties. Several plants in the United States are making coke from straight medium-volatile coal of 26 to 28 volatile content, and one at Rosita,, Mexico, from coal of 25 volatile. But no plants to my knowledge are using coal as low as 20 volatile. Since the Rosita coal appears to be a borderline coal from the angle of its expansion properties the coking of one of the straight lower volatile must be approached with caution.

5—There are few coals possessing any degree of coking properties which cannot be used in coke production by careful attention to its preparation and blending. The fact that coals of other types are available in this same region make improvement through blending very possible.

6—There are other workable methods of reducing iron ore other than the conventional coke-blast-furnace method. These will not be discussed here but it is known that their use has been considered. The technicians of not only Mexico but also of the other Latin American countries are keenly aware of their natural resources and their national needs. This paper emphasizes the fact that the Mexican technicians are working on their problem and attempting to speed the day of self-sufficiency for their country.

Salvatore Cortes-Obregon *(author's reply)*—I wish to thank Mr. Price for his kind remarks. The Mixtepec coal as shown in Table II has 30 pct ash and a free swelling index of 5, but when the same coal is washed to 15 pct ash it has a free swelling index of 8 to 9 and the volatiles increased from 17.4 to 20.7 pct.

A satisfactory coke has been produced from blends made in the Mexican laboratory using at least 40 pct of the Mixtepec coking coals with the other Oaxaca non-coking coals. Koppers in Germany report good coke obtained from the Oaxaca coal with a blend of 80 pct Mixtepec coal.

Consideration is being given the possibility of using methods other than the conventional blast furnace for the reduction of iron ore near the Oaxaca area; electric furnaces appear promising. The non-coking coals could be used to produce cheap electric energy and the coking coals to make metallurgical coke.

H — Industrial Minerals

Alkali Reactivity of Natural Aggregates in Western United States

by William Y. Holland and Roger H. Cook

DISCUSSION

Dexter H. Reynolds *(Chapman and Wood, Mining Engineers and Consulting Geologists, Albuquerque, N. M.)*—A number of questions are raised by conclusions and inferences made in the above-mentioned paper. The more troublesome of these concern use of the various pozzolans to combat the deleterious effects of the alkali-aggregate reaction.

The most alkali-reactive materials listed are opal and rocks containing opaline silica. The pozzolans mentioned specifically for use as amelioratives are opaline shales and cherts. These are stated to retard the expansion caused by the alkali-aggregate reaction. Another well-recognized pozzolan is diatomaceous earth, which consists principally of opaline silica.

A pozzolan presumably owes its effectiveness to its high reactivity with the alkaline liquid phase of the concrete mix. It appears reasonable to expect that finely divided opaline silica added as a pozzolan would be more susceptible to reaction with the alkalies present than would larger particles of the same material. The authors report that work with high and low alkali cements indicates that in the presence of alkali-reactive materials, deleterious expansion depends upon the alkali content of the cement. The total effect, therefore, should be more or less independent of the amount of reactive aggregate present, and still more independent of its state of subdivision. The deleterious effects should, if anything, be aggravated by the addition of a finely divided, highly reactive pozzolan.

Further, if the alkali-aggregate reaction is of great importance in the long-term soundness of concrete structures, the addition of a pozzolan to a concrete made with aggregate free from known deleterious materials would be a questionable procedure. The benefits reportedly accruing from such use of pozzolans are greater ultimate strength for a given cement content, increased resistance to deterioration by exposure to sulphate solutions and other mineral waters, and greater resistance to damage by wetting and drying and freezing and thawing. In view of the deleterious effects of highly reactive materials are these benefits ephemeral?

The same considerations apply to another alkali-reactive material, chalcedony, which appears to consist of ultrafine-grained quartz, with opal absent in detectable amounts. Quartz flour is notably reactive chemically and physiologically (cf. Ref. 11 of Holland and Cook's paper), a fact borne out by its effectiveness as a pozzolan, which presumably might be expected to offset the deleterious effects of the presence of chalcedony in the aggregate.

A second question of some importance concerns the reportedly highly deleterious reactivity of acidic and intermediate volcanic glasses, such as rhyolite, perlite, and pumice.

Air entrainment is listed as one of the ameliorative measures to combat the deleterious effects of the alkali-aggregate reaction. The *alkalic-silica* gel formed by the reaction may expand into air bubbles and thus not cause appreciable expansion of the concrete mass. It would appear then that pumice and perlite, particularly perlites of the pumiceous types and other types after expansion, would also tend to counteract the expansion, since these materials consist largely of voids and air bubbles. Certainly this would be expected of structural concrete in which pumice or perlite is used as total aggregate.

Finely ground pumice, perlite, and volcanic ash have been demonstrated to be active pozzolans (cf. *Pumice as Aggregate for Lightweight Structural Concrete* by Wagner, Gay, and Reynolds, Univ. of New Mexico Publications in Engineering No. 5, Albuquerque, 1950). In fact, the term *pozzolan* was first associated with finely divided pumice or volcanic ash. Such materials were used with hydrated lime as the sole cementitious agent in constructing public buildings, roads, and aqueducts by the ancient Romans. The deleterious alkali reactivity of the volcanic glass, itself containing several percent of the alkalies, apparently did not contribute to the remarkable state of preservation of those ancient structures, as exemplified by the Appian Way and the Pantheon Dome.

Still a third question involves the reactivity of constituents of concrete when exposed to various salt solutions. Resistance to deleterious expansion and cracking as a result of contact with mineral waters and its relationship to the mineral content of the aggregate are not mentioned by the authors. Yet the phenomena pictured in Fig. 1, and especially in Fig. 2, appear very much like those caused by exposure to mineral waters. The deterioration of concretes exposed to sulphate waters is generally considered related to the chemical constituency of the cement itself, particularly to the relative amount of tricalcium aluminate contained. Could not many of the ill effects presently blamed on alkali-aggregate reaction really have been caused by contact with sulphate or other salt-containing mineral waters? Or perhaps their use as mixing waters? May not the deleterious expansion be as much a function of the chemical makeup of the cement as it is of the mineral constituency of the aggregate? Would it not be just as important to use alkali-free mixing water as it is to use a low-alkali cement?

It appears obvious that resistance of cements and concretes to sulphate and other salt solutions cannot be left out of account in discussion of deterioration of concrete structures with time. This factor may be of equal or even greater importance than the alkali-aggregate reaction, particularly for concrete subjected to wetting and drying cycles, such as airstrip paving, water-retaining dams, and highway structures.

Another very important factor is called to attention on page 1022 of the article in *Mining Engineering*, October 1953, in that failure of concrete structures may result from poor construction practices and use of too high water-cement ratios. Both of these can contribute remarkably to decreased resistance to attack by sulphate waters, and presumably could have an equally remarkable effect upon extent of damage resulting from the alkali-aggregate reaction.

From the above remarks it appears that while alkali-aggregate reaction may be an important factor in decreasing the useful life of a concrete structure, it is not the only factor involved, and it may not be even a controlling factor. Likewise, many of the phenomena apparently associated with the alkali-aggregate reaction may have resulted from conditions which had little relationship to the alkali-reactivity of a constituent of the aggregate. Certainly if alkali-aggregate reactivity is a major factor in bringing about early failure, one cannot help feeling anxiety concerning the future of the many concrete structures in this country and abroad in which pumice and perlite were used as total or partial aggregates. This anxiety can only be dispelled by calling to mind that among the best-preserved relics coming down to us from ancient times are structures made with mortars containing highly alkali-reactive aggregates.

W. L. Merritt *(Western-Knapp Engineering Co.)*—My interest in the chemical reactions of the alkali in cements began in 1923 when I was superintendent of a diatomite plant at Terrebonne, Ore. Mr. O. Laurgaard, then city engineer of Portland, Ore., conceived the idea of adding pulverized diatomaceous earth to the concrete in a wall the city was building along the Willamette River. At that time the river was highly polluted with industrial wastes and sewage, and existing concrete structures were spauling badly. Mr. Laurgaard firmly believed this deterioration was caused by reaction between the excess alkali in the cement and the river solutions. He believed that the opal content in the diatomite would combine, if added in the mixer, with the available alkali in the cement, thus using up the alkali and preventing the reaction later, and so preventing spauling. This wall was so built; it is still in good condition and many carloads of diatomite for use as a concrete admix were sold. I supervised the plant test work from which we determined that 3 lb of diatomite per 100 lb of cement were required, which I believe, is still about the amount used when diatomite is used for a pozzolan.

My present approach to the problem of alkali-aggregate reaction is not neutralizing the effect in the mixer, but rather, removing the offending reactive materials in the aggregate by the use of heavy media separation, which is proving most satisfactory for this purpose. This takes advantage of the difference in specific gravity of the component materials in the aggregate.

In studying the specific gravities of the minerals now known to be very alkali-reactive, and thus detrimental in concrete, we find that many of them are comparatively light, see Table II.

Table II. Specific Gravities of Alkali-Reactive Rocks

Mineral or Rock	Avg Sp. Gr.	Mineral or Rock	Avg Sp. Gr.	*Na* or *K* Yielding Zeolites	Avg Sp. Gr.
Opal	2.10	Pitchstone	2.40	Analcite	2.3
Obsidian	2.31	Chalcedony	2.62	Natrolite	2.2
Ryolite	2.45	Chert	2.60	Chabazite	2.1
Dacite	2.45				

Most of the accepted desirable aggregate rocks have a specific gravity of from 2.6 to 2.7, so theoretically, aggregates treated in a heavy-media plant operating with a media of 2.47 gravity will be beneficiated by the removal of most of the light undesirable material.

The floatability of a fragment of rock is affected by its porosity, degree of soundness, and its shape. Porous, shattered, thin pieces are not desirable in concrete, so here again the aggregate is improved by removal of this type of material through gravity treatment.

Recent investigations have shown that in the sulphate soundness, resistance to freezing and thawing, and Los Angeles rattler tests, results have been improved on aggregates that were processed by heavy media . One test showed an 82 pct removal of chert and a 77 pct removal of soft material.

In the original paper it was brought out that the zeolites in an aggregate would release sodium and potassium in contact with the hydrating action in concrete, so that the alkali action would be increased when reactive materials were also present. Here again, as shown in the table, the offending minerals have a specific gravity of about 2.2 and can be removed from the aggregate by heavy-media treatment.

As Messrs. Holland and Cook have so ably pointed out in their original paper, no two gravel deposits could possibly be alike in composition. The diversity of source material and the complicated mechanics of their formation render each deposit a separate problem, and often a single deposit will vary greatly in composition both vertically and horizontally.

Heavy-media separation is not a cure-all but a process that will improve some aggregate deposits so they will meet rigid specifications that could not be met without such processing. A system of simple tests on the actual aggregate in heavy liquids will give a very close approximation of what can be done with heavy-media separation. Companies specializing in heavy-media separation systems, such as Western Machinery Co. and American Cyanamid Co., have facilities to make such tests and experienced engineers to interpret the results and make recommendations.

The economics of installing a heavy-media separation plant in a given location like any other mining process depends on many factors, including market, specifications, yardage, distance to a clean deposit, freight rates, and water supply. A well-designed plant handling 300 tph needs only one attendant. Power needs for 300 tph would be about 120 hp including that for conveyors into and out of the plant. Media loss can be held under 1 lb per ton or about two or three cents per ton of gravel. In a deposit with a very high percentage of *reactive* float material the cost of digging and disposing of the waste might become excessive.

In conclusion, while nature has provided some very satisfactory aggregate deposits, many others contain materials that are detrimental to concrete and will not meet rigid specifications. Such deposits must either be abandoned or beneficiated by any one of several approaches, such as 1—improved low-alkali cements, 2—several liquid and dry pozzolans, 3—air-entraining agents, 4—heavy-media separation treatment. The use of heavy-media separation on aggregates is a comparatively new approach, but the two or three plants now operating have been very successful in removing the objectionable material and producing a clean acceptable aggregate. An owner of a deposit that will not pass required specifications should investigate the application of heavy-media separation to his problems. An engineer who is responsible for the specifications on an important concrete structure should protect himself and the structure by investigating the aggregate deposit for alkali-reactive material and insisting on less than destructive percentages.

I should like to suggest that aggregates treated by heavy-media separation be given greater consideration in light of recent developments and the interesting possibilities its known capabilities can uncover.

William Y. Holland *(authors' reply)*—Many of the points raised in Mr. Reynolds' discussion pertain to pozzolanic and related reactivity and therefore do not bear directly on our paper. Some of these questions are discussed in the references cited or may be found in other references mentioned in these papers. It appears, however, that Mr. Reynolds has failed to recall that the fine state of subdivision of the genuine pozzolan as used with portland cement is the one property that assures the material will be beneficial. If the same material of which the pozzolan is composed were of a massive nature such as coarse sand grains or gravel, and occurred, let us say, as 15 pct of an otherwise satisfactory aggregate, it would undoubtedly react deleteriously with the alkalies of a high alkali cement. Although other factors are involved here, one of the principal reasons the coarse reactive material reacts deleteriously is that its reaction is slow, but highly concentrated in certain areas, and does not produce maximum expansion until some time months or even years after the concrete has hardened. On the other hand, the reactive material, if added as a fine powder (pozzolan), reacts rapidly with the alkalies and the reacting areas are very small and dispersed throughout the whole concrete mass, thereby producing, if at all, only insignificant pressures.

We agree that the pumiceous types of aggregates dispel at least some of the reaction pressures by absorbing the gel in voids and gas bubbles. However, if the total aggregate were pumice or perlite, as Mr. Reynolds suggests, their amount would be greater than the *pessimum* and there would not be any swelling in any case, as all the alkalies would be taken care of. A mix such as this would probably be a type of ordinary lightweight concrete. With this understanding, we agree about the Roman roads and aqueducts.

In regard to Mr. Reynolds' third question, we know, of course, that many concrete structures have failed or are failing because of attack of sulphate waters. The examples we chose, obviously, have suffered from alkali-aggregate reaction as determined either by us or by others at least equally competent. We do not favor alkali-aggregate reaction; in fact we could do very well without it.

In Mr. Reynolds' last paragraph, he states that alkali-aggregate reaction is not the only factor involved in decreasing the useful life of a concrete structure. We fully agree. There are probably only a very few concretes that we have examined, and we have examined many, where that statement would not apply excepting laboratory specimens and a few others. It is the degree, or the amount of deleterious effects, including design and formulation, caused by any one or a combination of the many destructive processes that limits the life of a concrete structure.

Mr. Merritt's discussion of the concrete wall constructed along the Willamette River, using diatomite with the mix, is very interesting. Of course, alkali-aggregate reaction may not have been involved here and the problem, therefore, does not bear directly on our paper. Mr. Laurgaard's contention that the diatomite takes up the available alkali is in line, however, with present day beliefs. There are other factors to be considered in the use of diatomite in concrete; some of these are increased plasticity, workability, water tightness, resistance to freezing and thawing, and unfortunately, difficulty with drying shrinkage.

Because of the complex mineralogical nature of most concrete aggregates, we have not tried the heavy media method of separation for these materials. It has been found that most of the deleteriously reacting materials are inherently a part of the individual pebbles of the aggregate and therefore are not amenable to this type of separation even though there may be a slight difference in the specific gravities. If an aggregate occurred in which the deleteriously reacting materials were in individual grains of different specific gravity, this type of separation might be beneficial.

Characteristics of Titaniferous Concentrates

by L. E. Lynd, H. Sigurdson, C. H. North, and W. Anderson

DISCUSSION

D. R. Grantham *(Institute of Mining and Metallurgy, London, England)*—This paper is a very valuable contribution to our knowledge of ilmenite and its alteration products. Two aspects are not treated in the paper and further information would be valuable.

How far are the alteration processes described influenced by the original composition of the ilmenite? The word *gangue* is used but it is not clear whether this refers to other chemical constituents in the ilmenite, e.g., magnesia, or to other minerals loose or in composite grains.

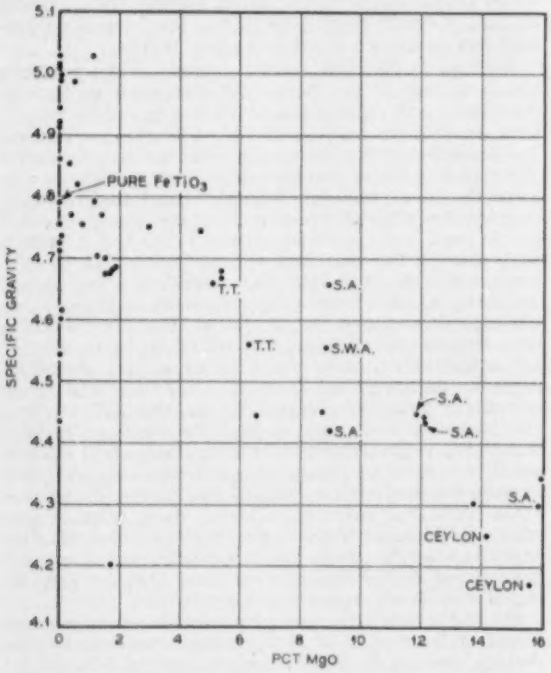

Fig. 17—Specific gravity of ilmenites. (After Grantham.)

Has the work described been related to specific gravities? Pure ilmenite is stated to have a specific gravity of 4.79. Some varieties range up to over 5.0, while high magnesic varieties have a specific gravity ranging to below 4.2. Alteration to hematite might lower the specific gravity and a porous texture due to the removal of iron and the formation of rutile might also lower it.

As specific gravity is a quick method which often gives an approximate indication of composition, data on the above would be valuable.

L. E. Lynd *(authors' reply)*—Because of the way in which beach sand deposits were formed, it is impossible to tell exactly what the original composition of the parent ilmenites might have been. Information on the influence of original composition on the alteration processes must be derived from other sources, perhaps by comparing the alteration products formed from ilmenite in place in a variety of rock deposits, where parent ilmenite and its alteration product can be recognized. No information of this type has yet been brought to our attention.

The material referred to as *gangue* includes all constituents other than oxides of titanium and iron. Nearly all of this was present as separate grains of garnet, augite, hornblende, quartz, feldspar, spinel, or other gangue minerals. Inclusions of gangue minerals within ore grains were rare, and we believe there is little MgO (1 pct or less) or other impurity actually held in the ilmenite structure in any of the concentrates studied.

No specific gravity measurements were made on the samples studied. However, the suggestion that alteration involving loss of iron would lower the specific gravity is a very reasonable one, and is in accord with the findings of Creitz and McVay (Ref. 9 of our paper) who reported two magnetic opaque fractions from Trail Ridge, Fla., to have the following specific gravities and compositions:

Sp Gr	TiO_2, Pct	Fe_2O_3, Pct
3.90 to 4.10	74.2	19.3
4.10 to 4.35	68.8	24.9

The specific gravity of this porous, high TiO_2 alteration product of ilmenite is closer to that of rutile (4.25) than to that of ilmenite. Due to the porosity, it may even be less dense than rutile, although lower in TiO_2 and higher in iron.

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MBD Golden Gate Meeting Attracts Crowd

More than 200 persons registered for the Fall Meeting of the Minerals Beneficiation Div. in San Francisco, September 24, at the Fairmont Hotel.

Responsible for the outstanding technical program held during the day was Norman Weiss, Regional Vice Chairman of the Division. John C. Lokken, Chairman of the San Francisco Section was the guiding spirit behind the well-planned physical arrangements for the meeting.

Speaking at the MBD Luncheon, AIME President Leo F. Reinartz discussed the current status of the Institute. Following him was Julian D. Conover, executive vice president and secretary of the American Mining Congress, who spoke on future AMC plans.

Will Mitchell, Jr., took over as toastmaster of the luncheon when it was learned that S. D. Michaelson, MBD Chairman, would be unable to attend because of family illness.

Registration, technical details, and luncheon arrangements were taken care of by G. Y. Murray.

C. F. Williams, metallurgist, Dow Chemical Co., and W. H. Reck, product manager, Western Machinery Co., were Co-chairmen of the morning technical session. Keith Kunze, mill superintendent, and Roy Nojima, chief assayer, Getchell Mine Inc., Red House, Nev., discussed colorimetric determinations of tungsten low grade ores. Norman Weiss detailed some of the innovations in design and flow sheet of the Silver Bell concentrator of American Smelting & Refining Co., 40 miles northwest of Tucson, Ariz. The 7500-tpd rated capacity copper flotation plant mills low grade ore from two open pit mines. Crushing is in three stages to ½-in., followed by single stage grinding in four 10-ft ball mills. A rougher concentrate is floated, reground, and cleaned.

A study of factors contributing to mud and pyroclastic ring formation, and methods for preventing and removing rings from rotary kilns were described by F. A. McGonigle and S. J. McCarroll, Manganese Inc., Henderson, Nev. In another paper presented during the morning technical session, M. C. Sutton, chief chemist, Calaveras Cement Co., San Andreas, Calif., briefed his audience on the instruments controlling an 11 ft 3 in.x360-ft cement kiln at his company. Feed rate, kiln speed, fuel flow, air fuel ratio, exit gas composition, and various drafts and temperatures are controlled and recorded automatically.

During the afternoon session L. G. Immonen, concentrator superintendent, Kennecott Copper Corp., McGill, Nev., and Neil Plummer, metallurgist, Kennecott Copper Corp., Garfield, Utah, were Co-chairmen. The first paper presented was an exposition of the theory and practice of solids flow through bins and feeders, with description of experimental methods, by Andrew W. Jenike, engineer, F. C. Torkelson Co., Salt Lake City. Mr. Jenike described the work done toward development of a quantitive method of design for flow of bulk solids in bins,

President Leo F. Reinartz addresses the luncheon of the Minerals Beneficiation Div. Fall Meeting in San Francisco. He covered topics of interest to Institute members.

More than 200 persons registered for the MBD fall meeting. Here is part of the huge crowd that filled the room for the luncheon that came between technical sessions held throughout the day. President Reinartz is speaking.

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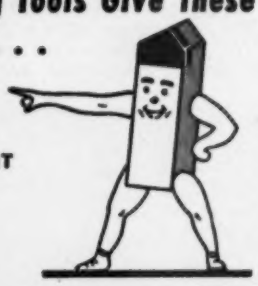

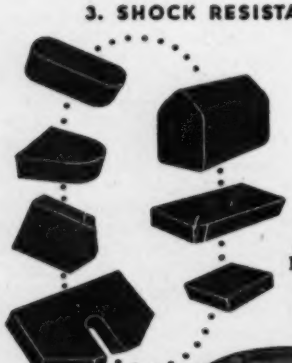

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MBD Meeting Cont'd

hoppers, chutes, and processing equipment.

J. E. Papin, mill superintendent, Phelps Dodge Corp., Morenci Branch, Morenci, Ariz., recounted the history of the development of the method used at Morenci in extracting salable molybdenite concentrate from normal copper concentrates. The presentation contained a brief delineation of methods currently practiced at other molybdenite plants and the research problems initially involved at Morenci.

Ball mill performance data, screen analyses, and operating costs pertaining to classification and grinding, when using a liquid-solids cyclone as a classifier in a closed circuit grinding operation treating bulk concentrate, were discussed in another paper. The authors, F. M. Lewis, superintendent of concentration, Tennessee Copper Co., Copperhill, Tenn., and E. C. Johnson, sales representative, The Dorr Co., Stamford, Conn., pointed out the advantages gained from replacing a sedimentation type classifier with the liquid-solids cyclone.

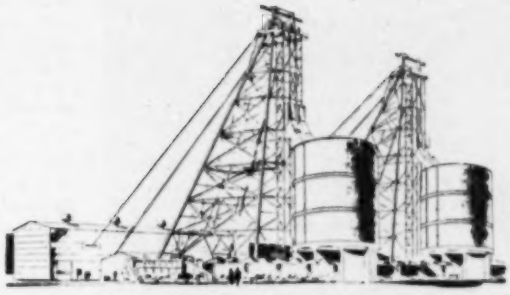
Percy Nicholls Award

John F. Barkley, chief, fuels utilization branch, U. S. Bureau of Mines, a 1935 member of the AIME, has been selected as the recipient of the Percy Nicholls Award for 1954. Presentation was made at the Fuels Conference in Pittsburgh at the end of October. The award is for "Notable Scientific or Industrial Achievement in the Field of Solid Fuels."

Establish New Section In Central New Mexico

An area of the United States which, except for Socorro County, had not been included in the territory of any Local Section, was formally included in the realm of the new Central New Mexico Local Section, established by the Board of Directors at the September 21 meeting. Its territory includes all of the counties of New Mexico not in the territory of the El Paso Metals, Carlsbad Potash, Permian Basin, or the proposed Hobbs Sections, specifically the following: Bernalillo, Colfax, Guadalupe, Harding, Los Alamos, McKinley, Mora, Quay, Rio Arriba, Sandoval, San Juan, San Miguel, Santa Fe, Socorro, Taos, Torrance, Union, and Valencia. At least four meetings will be held each year, with the annual meeting in January in Albuquerque. John A. Wood organized the new Section, whose first meeting was held on October 9.

Don't Forget . . .



1955 Annual Meeting

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American Institute of Mining and Metallurgical Engineers, Inc.

Ernest O. Kirkendall Named Administrative Assistant Secretary by Board of Directors

Having in mind the approaching retirement of Edward H. Robie as Secretary of the Institute, the Board

E. O. KIRKENDALL

of Directors, at the meeting on September 21, named E. O. Kirkendall as Administrative Assistant Secretary effective October 1. Dr. Kirkendall, since January 1946, has been Assistant Secretary of the AIME, also serving as Secretary of the Metals Branch and its constituent Divisions. He is 40 years old and a Michigander by birth. His bachelor degree was earned at Wayne University where he taught metallurgical and chemical engineering for ten years, and his master's and doctor's at the University of Michigan. While in Detroit he was employed on several consulting jobs.

The Board has recommended to the *new* Board taking office in February that Mr. Kirkendall at that time be made Secretary of the Institute, and that Mr. Robie be made Secretary Emeritus for the ensuing year, with duties to be assigned.

Since coming to the Institute he has been instrumental in organization of the Extractive Metallurgy Div. in 1948, the Metals Branch in 1949, and the inauguration of the JOURNAL OF METALS in 1949. Mr. Kirkendall served as editor of the magazine for the period preceding the hiring of a full-time editor. His work with the Iron and Steel Div. conferences is well known.

ASEE Lists New Slate of Officers

The Mineral Engineering Div. of the American Society for Engineering Education has announced the following new officers:

John C. Calhoun, Jr., Pennsylvania State University, chairman; M. D. Cooper, National Coal Assn., vice chairman; C. O. Frush, Iowa State College, secretary; and W. J. Rundle, University of Wisconsin, representative on General Council. These men also constitute the executive committee of the division.

AAAS 121st Meeting

The 121st meeting of the American Assn. for the Advancement of Science will be held at Berkeley, Calif., Dec. 26 to 31, 1954.

Section E, Geology and Geography, will present a program that has been divided into the following sessions: general geography; symposium on earth sciences from the air; smoker and vice presidential address; and general geology. Papers are invited but titles and abstracts of 250 words or less should be submitted first.

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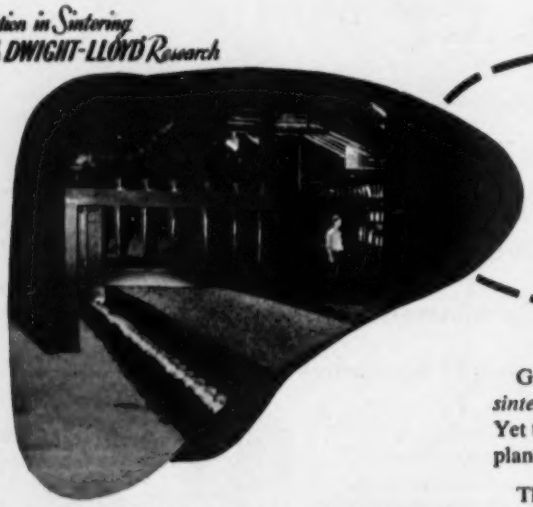

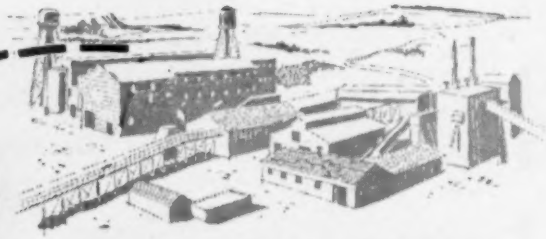

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Name Committee For Fairless Award

A year ago L. F. Reinartz announced to the AIME Board that a new award was being established to be presented annually to an outstanding man in the ferrous field. Last spring the U. S. Steel Corp. provided $13,000 as the principal of a fund to make the award possible, and it was decided that it would be known as the, "Benjamin F. Fairless Award in Iron and Steel Production and Ferrous Metallurgy."

Rules of the award have now been approved by the Board. Briefly they are as follows: The award shall be a bronze plaque. There are no limitations regarding nationality or membership in the Institute. The candidate must be a living person, willing to be present in person to receive the award. A person who has been awarded one of the other major awards of the Institute is ineligible. Nine members of the AIME constitute the award committee. The procedure of selection is essentially the same as for the other major awards of the Institute (details on request from the Secretary's office); nominations must be made by the committee but anyone can suggest suitable names to the committee.

The names of the committee which will select a candidate for the award for the presentation at the Annual Meeting of the Institute in February 1955 are as follows, for terms expiring as indicated: L. E. Berner, Chairman, 1955; E. C. Smith, 1955; K. C. McCutcheon, 1956; J. N. Marshall, 1956; J. S. Marsh, 1957; H. W. Graham, 1957; John Chipman, 1958; G. B. McMeans, 1958; L. F. Reinartz, President AIME, ex officio, 1955.

Half of Metals Branch Research Fund Received

Approximately half of the $50,000 goal for the Metals Research Publication Fund had been received by the end of September. At its September meeting, the Board of Directors authorized a levy on this fund sufficient to publish approximately 868 pages of Metals Transactions material this year, as had originally been planned, without increasing the Metals Branch deficit beyond the $29,000 figure set in the revised budget. Otherwise a sharp reduction in the amount of metals research papers would have been necessary. Some $7600 will be used from the new Fund for this purpose. The Board also authorized publication of an extra 144-page Transactions supplement in January 1955, the cost of which would be borne by the Fund.

The objective for this new potash plant was AS NEARLY AUTOMATIC OPERATION AS POSSIBLE, throughout every phase until the quality-controlled finished product is shipped.

3,000 tons of feed per 24 hours was the capacity sought, and every aspect of production and treatment was scaled to that figure.

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Announce Rules For Robert Peele Award

AIME Directors have approved the rules for the Robert Peele Award, which was established last year to honor the distinguished mining engineer, educator, and author for whom it was named. The award will be given for the most outstanding paper by a younger member of the AIME on a subject in the field of mining, geology, or geophysics, published in the Transactions within a two-year period ending May 1 of the year preceding the award. In the case of joint authorship, one or more authors must be AIME Members. Papers will be judged for (1) significance of the contribution to the engineering literature of metal mining, geology, or geophysics; and (2) excellence of technical writing.

Primary financing of the award will be by MINING ENGINEERING but a permanent fund is to be raised, the income from which should be at least $300 a year.

The first award will be made at the Annual Meeting of the AIME next February.

Change To Be Made In Presidential Rotation

Normally, with the usual rotation, a mining man would be destined to be President of the AIME in 1959. However, that will be the year of the 100th anniversary of petroleum industry in this country and therefore it would be appropriate for the president of the Institute to be representative of that industry. The Mining Branch Council has given its assent, provided a Mining man would be selected for the presidency in 1960 and 1961. The rotation will therefore be: 1955, Mining; 1956, Petroleum; 1957, Mining; 1958, Metals; 1959, Petroleum; 1960 and 1961, Mining; 1962, Metals. The president, of course, serves as president-elect in the preceding year.

SEG to Meet At Dallas, Texas

The 8th Annual Midwestern meeting of the Society of Exploration Geophysicists is scheduled for November 18 and 19 at the Adolphus Hotel, Dallas.

Registration will begin November 17 for more than 1000 exploration men from Texas, Louisiana, Oklahoma, and New Mexico. Members of the Dallas Geophysical Society head the meeting committees, assisted by members of five other participating SEG local sections in Fort Worth, Midland, Shreveport, Oklahoma City, and Tulsa.

Around the Sections

• Theodore B. Counselman, AIME Vice President and division manager of The Dorr Co., spoke before the **Tri-State Section** at Joplin, Mo., on the need for employers and the engineering profession to devise means for attracting young men. He noted that current output of engineers by colleges is only 19,000 per year against a needed 50,000 engineering replacements. Mr. Counselman also discussed Institute affairs from the standpoint of the New York office and the national organization at a luncheon earlier in the day.

• A joint meeting of the **Intermountain Subsection** and the **Colorado Plateau Section** was held last month at the Denver Hotel, Glenwood Springs, Colo. An afternoon conference was held in the Riverside Room with E. M. Paris in charge of program arrangements. Cocktails and dinner in the Capri Room followed.

• Last meeting of the **Black Hills Section** started with a field trip to the Colony plant and pits at Colony, Wyo. The trip was conducted by R. K. Collins, of the Baroid Div., National Lead Co. After the field trip C. G. Scott, also of the Baroid Div., spoke to the members of the section on bentonite. J. P. Gries has been appointed Student Counselor for Student Associates.

• Fifty members of the **Lehigh Valley Section** joined together at the Hotel Traylor, Allentown, Pa., for the section's fall dinner Meeting. New officers nominated for the coming year are: A. C. Most, Jr., Chairman for one year; A. F. Peterson, Jr., Vice Chairman; and J. L. Rodda, Manager, both for three years. R. K. Waring, assistant manager of research, New Jersey Zinc Co. of Pennsylvania, spoke on *Recent Progress in Zinc Smelting*. The section will hold its annual dinner meeting December 10 at the Hotel Bethlehem, Bethlehem, Pa.

• The new metallurgical laboratories of the Missouri School of Mines were toured by members of the **St. Louis Section** in conjunction with the visit of FOA Mission 228, a group of European metallurgists. Following the tour these papers were presented: *Mineral Dressing Research in Educational Institutions*, by A. Legsdin, associate professor of metallurgical engineering; *Mineral Dressing Research in the U. S. Bureau of Mines*, by M. Fine, metallurgist, USBM, Rolla; *Mineral Dressing Research in Industrial Concerns*, by T. M. Morris, associate professor of metallurgical engineering; and *Recent Progress in Hydrometallurgical Treatment of Low Grade Ores*, by A. W. Schlechten, professor of metallurgical engineering. A round table conference in the afternoon covered *Desirable Objectives for Future Mineral Dressing Research*. A. W. Schlechten and P. G. Kihlstedt of the Royal Institute of Stockholm presided.

• J. B. McMorran, chief engineer, New York State Power Authority, described plans for power generation on the St. Lawrence River to members of the **Adirondack Section** at a regular monthly meeting. He outlined construction needed to produce power by 1956 and the interrelation of the development to the St. Lawrence Seaway. The meeting was held at the Gouverneur Country Club, Gouverneur, N. Y.

• Richest "pay streak" in the Mother Lode of California was not the Kennedy, the Argonaut, or Carson Hill, but the limestone of Calaveras County. Calaveras Cement Co. has been mining this limestone deposit since 1926, producing more than $70 million in new wealth. The Calaveras story was told to members of the **San Francisco Section** by E. M. Barker, the firm's operating vice president. He covered rock drilling, milling, and burning improvements in the last six years, enabling Calaveras to triple production capacity. A color movie showing the company's operations followed.

• F. W. Galbraith, head of the dept. of geology and mineralogy, University of Arizona, addressed members and wives of the **Morenci Subsection** on his travels in French Morocco. He used color slides taken during his travels to illustrate his talk.

• **A new film on drilling and blasting operations in an underground limestone quarry has been released by Gardner-Denver Co., Quincy, Ill. The film is in color and has a sound track. Called $CaCO_3$, the film takes its audience on a conducted tour through the 38-acre underground quarry of the Black White Limestone Co. in Quincy. It shows the 2-man, 4-drill rig putting in 15-ft holes in the 25x50-ft face. Loading of holes and the actual blast are shown. Removal of rock, crushing and grinding, and some of the end uses of the material are traced.**

Around the Sections

The joint meeting of the Nelson and Trail Branches of the Canadian Institute of Mining and Metallurgy and the **Columbia Section of the AIME** was held at the Canadian Exploration Ltd. site on Iron Mountain, 7 miles southeast of Salmo, B. C. Some 40 men of this company, led by J. D. Simpson, vice president, G. A. Gordon, general manager, and G. W. Walkey, assistant general manager, entertained some 100 visitors, more than 40 of whom were AIME members and their guests. Three tours of about 1½ hr duration were organized to show the different phases of operation. From the assembly point at the main camp trucks took the party to the starting point of the three trips.

The mine workings extended from 3800 to 4400 ft above sea level. Iron Mountain itself is 5200 ft above sea level. It was originally prospected for gold and the pyrites—fool's gold—could be seen easily. Between 1907 and 1926 small shipments of lead ore and lead concentrates were made. Scheelite was discovered between 1938 and 1941 and because of the war activity it was developed. By 1942 a 300 tpd concentrator was built. Since then more tungsten ore has been found and the operation has been taken over by Canadian Exploration Ltd. Operations on lead and zinc have been increased also by trackless mining.

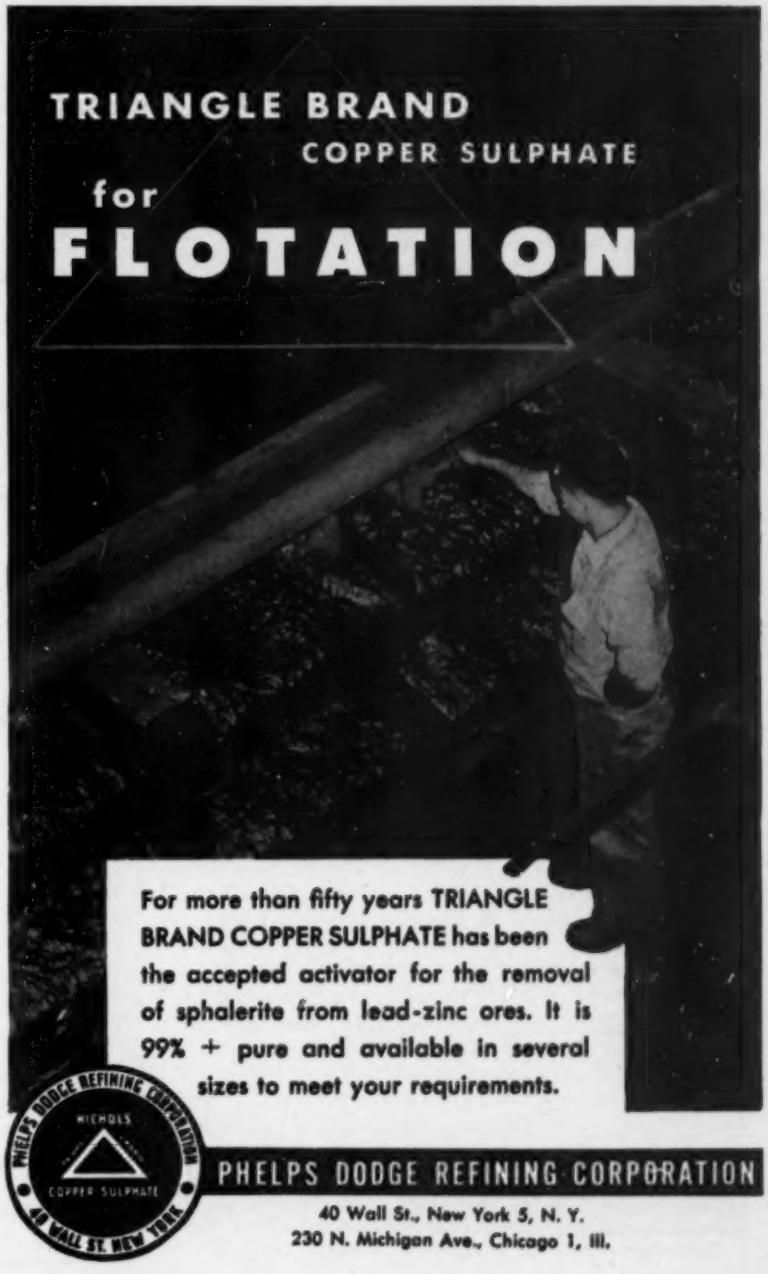
Each party of five was escorted by an engineer or foreman. The lead-zinc trackless mining operations were particularly impressive. The geologists on the tour had a field day examining the walls and the rocks.

The underground crushing plant, the upper end of the ore transportation system, and the incline shaft collared on the 3800 level at present being developed were visited. The 3800 crosscut, entry to both the incline shaft and the crushing plant, provides a typical geological cross section of the Emerald-Tungsten trough. Scheelite was evident under black light.

The third operation shown consisted of milling, grinding, and flotation in the tungsten mill. The final product, 65 pct or better tungsten oxide, is shipped to the U. S.

Soon after 4:30 all groups returned, ready for refreshment. At 6:00 a group of ladies of this camp provided the meeting with an excellent turkey dinner. After a toast to the Queen and the President, welcome and introductions were made by W. Peck, vice president of the Nelson Branch of the CIM, Mr. Kinney, Chairman of the Columbia Section of the AIME, and Mr. Marr for the Northwest Mining Assn. Mr. Simpson, vice president of the Canadian Exploration Ltd., welcomed the group and outlined a few experiences of that company and then showed some 16mm color movies of his company's gold mining operations in New Guinea and other movies of a trip along the Amazon which he had made with a partner in 1930.

Soon after this, the groups dispersed, but not before requesting another joint meeting next year. The efforts of Vince Killeen, Secretary of the Nelson Branch of the CIM, and the organization of the tours by the Canadian Exploration Ltd., left nothing to be desired. Spontaneous and sincere rounds of applause expressed the visitors appreciation.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Post-Annual Meeting Trip Promises Winter Fun

Some of the best skiing in the world awaits AIME members who journey to Iron Mountain, Mich., for the Post-Convention Snow Carnival. The famed winter resort will be all decked out in white for Institute members who thrill to the cry of "TRACK"—like to ice skate—or go in for the rugged sport of ice fishing.

The specially arranged trip starts February 18 from Chicago, right after the Annual Meeting windup. Train-time on the Milwaukee Road is 12:30 pm, with arrival at Iron Mountain scheduled for 7:45 pm. The first morning will be devoted to individual activities. Iron Mountain slopes and trails are graduated from beginner's hills to expert trails testing the mettle of the best of skiers.

One of the most popular winter activities at the resort is ice fishing. Arrangements have been made for the areas best guides to take AIME members to the best spots in the region. And if you don't have your own skis, skates, or ice fishing gear along, don't worry. Arrangements have been made to equip Institute members with everything they need. However, it is recommended that you bring your own skates.

Saturday afternoon will see some of the finest ski jumpers in the world take off down the Iron Mountain jump in one of the season's most important tournaments. Ski jumpers are a breed of men apart, ripping down the slide at breakneck speeds and leaping out into space in an attempt to outjump each other for medals, marbles, or just fun.

During the evening a gala dance and dinner has been scheduled. Another attraction of the resort is sleigh riding, complete with bells, horses, and heavy blankets to keep out the cold.

Sunday morning can be used for relaxation or another round of winter sports. A skating exhibition and contest has been scheduled for the afternoon. Departure is slated for 11 am Monday by train for Chicago or home.

Cost of the entire week end has been pegged at $65 per person double occupancy. Accommodations for 200 people have been arranged for at the Hotel Dickenson and several modern, clean, and comfortable motor courts. Lodgings, all with private baths, will be assigned on a first come, first served basis. For reservations and further information write to: J. H. Theilig, president, Theilig International Tours Inc., 119 S. State St., Palmer House, Suite 217, Chicago 3, Ill.

Divisions May Name Nuclear Committees

With a constantly growing number of AIME Members interested in one or another of the phases of the generation and use of nuclear energy, much interest has been expressed in the proposed all-Institute committee on this subject. It is now felt that the most practical method of recognizing Members' interest in the subject is through the appointment of divisional committees, each of which would develop papers and arrange for sessions in the specific field of the division. Such committees would in all likelihood work jointly to a considerable extent, with each other and with similar committees of other societies. Sessions involving various aspects of the finding and utilization of uranium and the other metals of possible similar use may be arranged for the Annual Meeting of the Institute, and for a potential meeting with other societies currently being arranged by Engineers Joint Council the date of which is tentatively set for July 11 to 16, 1955.

The chairmen of the various divisional AIME committees on nuclear energy might compose an all-Institute committee for joint action.

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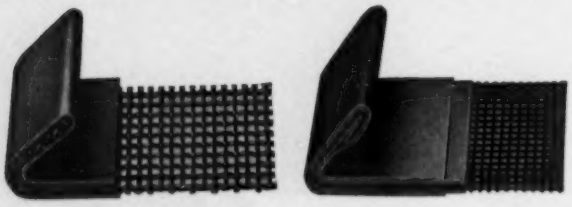

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Personals

F. B. SPEAKER

Francis B. Speaker has been named manager of nonferrous metal mining industry sales development by Hewitt-Robins Inc., Stamford, Conn. Mr. Speaker, formerly chief of the Nonferrous Metal Section Mining Div., War Production Board, joined Hewitt-Robins in 1945. From 1951 until 1953 he returned to government service as director of the Mining Div., Defense Materials Procurement Agency. He is at present consultant to the General Services Administration.

Robert F. Griffith, who was with the U. S. Bureau of Mines, Washington, D. C. is now superintendent of Marine Minerals Inc. This company is undertaking a bucket-line dredge operation in Aiken County, S. C., for the recovery of titanium and other heavy minerals. A separation plant will be constructed in connection with the dredging operation.

Charles H. Strange of Kiln-Gard, maker of automatic kiln cut-offs, has moved from White Plains, N. Y., to Jacksonville Beach, Fla.

David D. Baker is the new staff engineer in the office of **E. R. Gordon,** director, Exploration Div., Atomic Energy Commission, Grand Junction, Colo. Mr. Baker will act in an advisory capacity and provide liaison with the mining and processing divisions. Mr. Baker recently was with the Utah Construction Co. as manager of a contract between this firm and the Korean Government for the rehabilitation of Korean tungsten mines. Mr. Baker has also been employed by Union Mines Development Co., Pacific Mining Co., Alaska-Juneau Co., U. S. Vanadium, Big Horn Mining Co., and American Smelting & Refining Co.

George V. Keller, geophysics branch, U. S. Geological Survey, Washington, D. C., has been transferred to Denver.

Thorsten M. Andersson has joined Neptune Gold Mining Co., Bonanza, Nicaragua, as assistant mill superintendent. Mr. Andersson left Saudi Arabia in August when the Saudi Arabian Mining Syndicate Ltd. closed down operations at the company's property at Mahad Dahab, Saudi Arabia.

Elton A. Youngberg has been named director of the Mining Div. of the Grand Junction, Colo., operations office of the Atomic Energy Commission. Before joining the AEC in 1952, Mr. Young was with the U. S. Geological Survey as geologist in charge of the Juneau, Alaska, office and survey member of the Defense Minerals Exploration Administration field team that administered the DMEA activities in Alaska.

George A. Parks, formerly student and teaching assistant, University of California, Berkeley, is now at MIT where he is a student and research assistant in the dept. of metallurgy.

John F. Hatch, III, is the first recipient of the newly established Mineral Engineering Fellowship at Harvard Graduate School of Business Administration. Mr. Hatch, supervisor of one of the mines of International Mineral & Chemical Corp., Bartow, Fla., is a 1949 graduate of Colorado School of Mines.

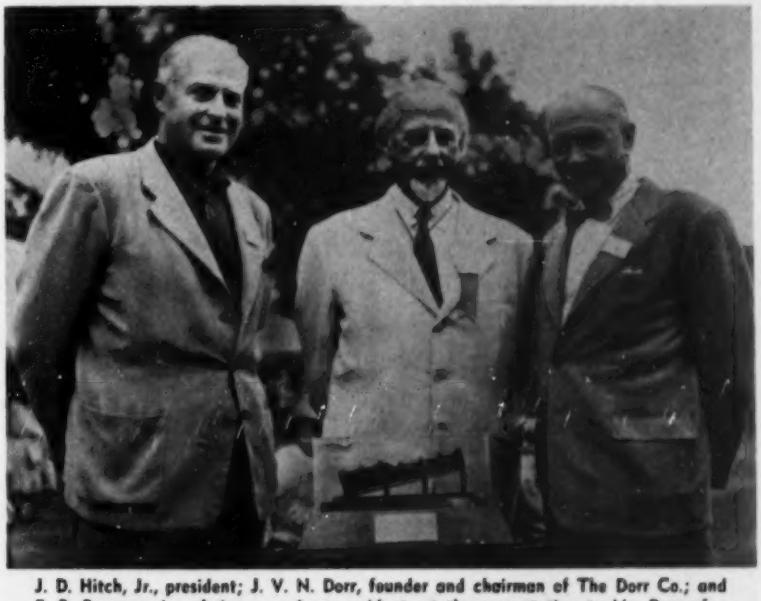
P. V. G. Ford, general manager of mines, Sinai Mining Co. Ltd., Port Tewfik, Egypt, has been promoted to deputy managing director of this company with residence in Cairo. Mr. Ford recently returned from a vacation in England and Cyprus. In Cyprus he visited the Amiandos asbestos mine and a small chrome mine.

Kenneth H. Gibson has joined the staff of Roger Williams Technical & Economic Services Inc., New York, as an economic geologist. He was a member of the technical economics group of Arthur D. Little Inc., Cambridge, Mass., and before that assistant to the state engineering geologist of New York State.

Burton L. Ballard, sales engineer, Ingersoll-Rand, Salt Lake City, has been transferred to New York as a special representative.

Stoddard S. Burg, research assistant, mining dept., Pennsylvania State University, State College, Pa., is now an industrial engineer, U. S. Steel Corp., Lynch, Ky.

Norman R. Alston, senior metallurgist, Garfield Smelter, American Smelting & Refining Co., Garfield, Utah, has been named superintendent of smelters, Braden Copper Co., Rancagua, Chile.



J. D. Hitch, Jr., president; J. V. N. Dorr, founder and chairman of The Dorr Co.; and E. R. Ramsey, vice chairman and ex-president, at the presentation to Mr. Dorr of a scale model of the first Dorr classifier in celebration of the 50th anniversary of its inception. Invented in 1904 to solve a milling problem at the Dorr, Lundberg & Wilson mill in the Black Hills of South Dakota, it was the first of the three basic inventions upon which the company was founded.

ROBERT W. MAHONEY

Robert M. Mahoney has been appointed assistant to **C. H. Hageman,** manager, industrial relations dept., Union Carbide & Carbon Corp., New York. Immediately following graduation in 1937 from California Institute of Technology, Mr. Mahoney joined U. S. Vanadium Co., Div. of Union Carbide & Carbon Corp., at Bishop, Calif. He became manager of industrial relations for this company in 1941 and was later promoted to assistant manager of industrial relations for Electro Metallurgical Co., one of the principal divisions of Union Carbide.

B. R. Coil has been advanced from assistant general manager to manager of Miami Copper Co. and Copper Cities Mining Co. Mr. Coil first joined Miami Copper Co. in 1938 as mine engineer and later became mine superintendent at Castle Dome Copper Co.; general superintendent of that company; general superintendent of both Miami and Castle Dome, and then assistant general manager.

A. F. Banfield and **Roland F. Parks** of Behre Dolbear & Co., New York, have been in Labrador doing field work in connection with a valuation of iron ore deposits.

Alton K. Guard, formerly project engineer, Morrison-Knudsen Co. Inc., Soda Springs, Idaho, is a geologist with Boyles Bros. Diamond Drillers, Salt Lake City.

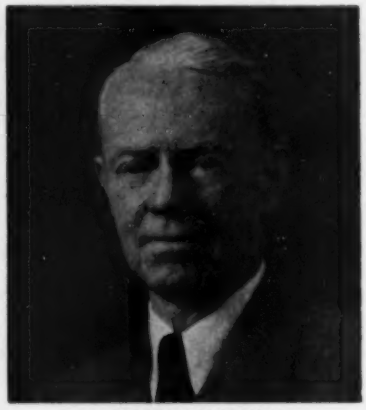
F. A. Barrón, formerly chief, mine section, Banco Minero del Peru, Huachocolpa, Huancavelica, Peru, and then safety engineer, Quiruvilca mine, Northern Peru Mining & Smelting Co., is now section mine foreman at this same mine.

Frank E. Seery, with the General American Transportation Corp., has been transferred from Pittsburgh to Chicago.

L. H. Lange, vice president, The Galigher Co., Salt Lake City, recently returned from a round-the-world trip of approximately six months. Mr. Lange spent nine weeks at Mt. Isa in Australia as consulting metallurgist for Mt. Isa Mines Ltd. After visiting other important mining areas there—Broken Hill, Radium Hill, King Island, and the Kalgoorlie gold district, he flew to South Africa. Being consultant for Tsumeb Corp., he visited this base metal operation in South West Africa and then visited the O'Okiep copper mines, the copper belt in Northern Rhodesia, and several pyrite flotation plants in South Africa. Mr. Lange returned to the U. S. via England, stopping off at Buchans, Newfoundland, for American Smelting & Refining Co.

Carl Tolman has been named vice chancellor and dean of faculties, Washington University, St. Louis. He has been acting in this capacity since June 1953. Mr. Tolman has been associated with the university as a member of the faculty and in various administrative posts since 1927. From 1942 to 1945 he was on leave as chief of the quartz crystal-abrasives section, Mineral Div., FEA, Washington, D. C. As a consulting mining geologist, Mr. Tolman has led geological surveys in the West, Canada, and Mexico.

Ralph I. Clark, mining geologist, is with the Pacific Northwest Area, Gladding, McBean & Co., Seattle. He was with San Luis Mining Co., Tayoltita, Durango, Mexico.



R. GORDON WALKER

R. Gordon Walker has retired as vice president and general sales manager, Oliver United Filters Inc., New York. Following graduation in 1908 from the College of Mines, University of California, Mr. Walker was employed by the North Star Gold Mines, Grass Valley, Calif. There he became closely associated with the first continuous vacuum filter, an invention of **Edwin L. Oliver,** also at the North Star, who later founded Oliver United.

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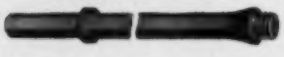

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E. P. Flint, chemist and chairman, ceramics & minerals dept., Armour Research Foundation, Chicago, is now with Mallinckrodt Chemical Works, St. Louis.

F. F. Espie, general superintendent, Western Mining Corp. Ltd., Kalgoorlie, Western Australia, was awarded the Institute Medal by the Australasian Institute of Mining and Metallurgy in June. **M. A. Mawby,** president of AIMM, made the presentation. The citation accompanying the medal reads: "In recognition of his professional work in Burma and in Western Australia and his leadership in the safe transference of the Burma Corporation staff in 1942 through Japanese-occupied territory." Mr. Espie has been a councillor of AIMM since 1949 and was president in 1948.

ANDREW ROSTOSKY

Andrew Rostosky, for the past six years Eastern district manager of advertising sales for Mechanization Inc., is now publisher and editor of *Mining and Quarrying.* The first issue of this product news tabloid appears this month. Issued by Ros-Mac Publishing Co., New York, this monthly will serve the solid minerals industry.

Edward A. Loria has been appointed a staff metallurgist at the central metallurgical office, Crucible Steel Co. of America, Pittsburgh. He was formerly associated with the research & development dept., Carborundum Co., Niagara Falls, N. Y., as senior metallurgist.

James W. Cole, formerly metallurgist with American Cyanamid, Winchester, Mass., is now mill superintendent for Surcease Mining Co., Johannesburg, Calif.

Robert M. Hurst, who was with Miami Copper Co., Miami, Ariz., is a partner in Technical Services Inc., Grand Junction, Colo. This firm does engineering and geological work for independent uranium miners on the Colorado Plateau.

James Howard Terry is assistant geologist, Cerro de Pasco Corp., Cerro de Pasco, Peru. Mr. Terry was a cartographic aid for Inter-American Geodetic Survey, the headquarters of which are in Panama. He worked as a triangulation observer on the coast and also in the Andes region of Peru.

B. E. Grant, U. S. Smelting, Refining & Mining Co., **Miles P. Romney,** manager, Utah Mining Assn., and **Clark L. Wilson,** New Park Mining Co., have been named to head Salt Lake City Chamber of Commerce committees. Mr. Grant heads the committee on education, Mr. Romney on mining, and Mr. Wilson, on natural resources.

A. M. Gaudin, professor at MIT, Cambridge, Mass., attended the World Conference on Detergency and Surface Active Agents in Paris, August 30 to September 2, and presided over the session on flotation.

Frederick R. Gruner, formerly with Allis-Chalmers Mfg. Co., Milwaukee, has been appointed vice president in charge of operations, Warren Foundry & Pipe Corp., New York.

Arthur L. Bailey is sales engineer for Mine Supply Co., Carlsbad, N.M. He was with Joy Mfg. Co., Huntington, W. Va.

JAMES BOYD

James Boyd, exploration manager, Kennecott Copper Corp., New York, and former director of the U. S. Bureau of Mines, is scheduled to address the 8th annual Midwestern meeting of the Society of Exploration Geophysicists November 18 in Dallas.

Hugh M. Craigie is president and general manager, Sabre Uranium Corp., Grand Junction, Colo. Mr. Craigie was assistant general manager, San Francisco Mines of Mexico Ltd., San Francisco del Oro, Chih., Mexico, and had been with this company for 20 years.

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J. W. BLOOMQUIST

J. W. Bloomquist has been appointed to the position of domestic sales manager, Euclid Div., General Motors Corp., Cleveland. Mr. Bloomquist has served as manager of the Hibbing, Minn., factory branch since 1950. He joined Euclid in 1945 as district sales representative covering northern Michigan, Minnesota, North and South Dakota and was assistant manager of domestic sales prior to assignment as Hibbing branch manager.

Robert W. Gerling is with Volcan Mines Co., Ticlio, Peru.

James L. Taylor, formerly chief, project construction, International Minerals & Chemical Corp., Chicago, is now an engineer with Oliver Iron Mining Div., U. S. Steel Corp., Virginia, Minn.

John B. Botelho, mining geologist, has accepted a position in Brazil as field engineer with American & Foreign Power Co. on the Peixoto project. He was field engineer with Bateson & Stolte, Oakland, Calif.

Claiborne C. Van Zandt has been elected vice president in charge of operations and engineering, Long Star Cement Corp., New York. He was formerly vice president and chief engineer. Mr. Van Zandt will be succeeded as chief engineer by **Roy A. Rasmussen.**

S. Broersma is mechanical superintendent, East Malartic Mines, Marlartic Norrie, Que. Before returning to Holland in 1953, Mr. Broersma was mill superintendent, Patino Mines & Enterprises C. I., Catavi, Bolivia.

A. M. Davidson is employed as an engineer with Lecco Machinery & Engineering Co., Bluefield, W. Va.

C. R. Amis is assistant unit superintendent, American Smelting & Refining Co., Aurora Xichu, Gto., Mexico.

JOHN L. BOARDMAN

John L. Boardman, chairman, bureau of safety, Anaconda Copper Mining Co., Butte, Mont., is chairman of the national campaign to reduce falls of ground accidents in mines. This campaign to be launched Jan. 1, 1955 is sponsored by the Mining Section of the National Safety Council, Chicago.

James K. Richardson is assistant general manager in charge of labor, public, and industrial relations at Kennecott Copper Corp.'s Chino Mines Div. Mr. Richardson, who was assistant to the general manager of Kennecott's Utah Copper Div., joined the company in 1950 as industrial relations engineer for the Western Mining Divisions. Prior to that he had been manager of the Utah Mining Assn.

Charles A. Beckman, formerly senior engineering aide, New York State Dept. of Public Works, Bureau of Soil Mechanics, Lathams, N. Y., is now studying at the University of California, Berkeley, for his M.S. in geology.

Roy B. Young, plant superintendent, Quebec Iron & Titanium Corp., Montreal, has been appointed general manager to succeed **F. E. Walling,** who has resigned. Mr. Walling is now with Emhart Mfg. Co., Hartford, Conn.

A. H. Barrios has accepted the position of director of employee relations with Cerro de Pasco Corp., Lima, Peru. He was with Andes Copper Mining Co., Potrerillos, Chile.

Heath Steele, who is retiring at the end of this year as a vice president and a director of American Metal Co. Ltd., is also retiring from the various positions he has held in subsidiary companies. Among these is the Southwest Potash Corp., a wholly owned subsidiary. Mr. Steele has been president of Southwest Potash since its inception. **Thomas W. Childs** has been elected to succeed him as president.

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Obituaries

Thomas W. Guy (Member 1926) of Charleston, W. Va., died Sept. 11, 1954 after a brief illness. He was a nationally known preparation engineer. "Although he designed and supervised construction and operation of numerous coal production plants, Mr. Guy's greater contribution to the industry was his leadership in development of improved methods for handling and preparation of coal for marketing. Mr. Guy was for many years active in the affairs of the American Mining Congress Coal Preparation Committee and for five years served as chairman of the group. Through his work on the committee and by his many technical papers the coal industry has benefited by improved techniques he developed."

Mr. Guy was born in Nottoway County, Va., in 1884 and received his practical training under his father, a graduate engineer, and T. M. T. McKennan of Chase City, Va. He was chief engineer of Clinchfield Coal Corp. from 1911 to 1917; served as an engineer with the 25th Engineers during World War I and on discharge was made chief engineer and then general superintendent of Boone County Coal Corp., Sharples, W. Va. In 1929 Mr. Guy opened an engineering office in Charleston and specialized in preparation. He was associate editor of *Mechanization* from 1937 to 1944 and the author of many technical reports, discussions, and short articles published in the past 26 years by the American Mining Congress, the AIME, *Mechanization*, and others. Mr. Guy was chairman of the AIME Coal Preparation Committee in 1947.

John H. Leavell (Member 1907) of Tulsa died Sept. 15, 1954 in Queen's Hospital in Honolulu where he was on a vacation with his wife. He had suffered a stroke three weeks before. Mr. Leavell was president of Leavell Coal Co., Tulsa, and president of Trojan Coal Co. and a director of Kirk Smelters of California. In his tribute to him, the noted artist, author, and war correspondent, Tom Lea wrote: "There was power in Leavell. He was granted with long life, full of action; he was gifted with a sensitive mind, full of perception. That blend of action with perception gave his character its quality and his life its meaning."

Mr. Leavell, better known in recent years as Colonel Leavell, was born in Georgetown, Texas, in 1883 and graduated from MIT in 1907. His early experience was gained with the USGS in the Black Hills, as an assayer with Big Bend Cinnabar Mining Co., Brewster County, Texas, and with the Nevada-Utah Mines & Smelters Corp. as an assayer and engineer. He was also a consulting engineer for the Campbell Interests of Texas in Mexico City. During World War I he was a major with the 316th Engineers. He was awarded two corps citations, a French Croix de Guerre with Bronze star, and the Distinguished Service Cross. He was in charge of public utilities in the U. S. Zone of Germany during the postwar occupation and served on the Interallied Coal Commission. Mr. Leavell went to Tulsa in 1920 to develop coal mining interests. During World War II he was called on by the State Dept. to serve as petroleum attaché and was promoted to colonel. In recent years Colonel Leavell had devoted his time to mining and petroleum interests and foreign travel.

Rush T. Sill
An Appreciation
by Donald M. Baker

Rush T. Sill (Member 1944), consulting engineer of Los Angeles and a partner in the firm of Ruscardon Engineers, met his death in an airplane crash near Calmilli in Lower California, about 300 miles south of the American-Mexican border, on Aug. 19, 1954. Sill was making a reconnaissance for the National Bulk Carriers of New York to locate a site for a breakwater on Viscaino Bay, and a quarry site, where ships could load salt from a large deposit east of Scammons Lagoon.

Sill was born on Nov. 27, 1882, near Fremont, Neb., and received the degree of Engineer of Mines from Colorado School of Mines in 1906, where he was a star on the football team.

His first work was as assistant geologist for the Great Northern RR in the Mesabi Range, at Hibbing, Minn. This was followed by two years, from 1907 to 1909, as mining engineer in charge of iron mines for this railroad in the Mesabi Range.

From 1909 to 1911 he served as chief engineer and general manager for the Sinaloa Sonora Exploration Co., working on the west coast of Mexico.

From the latter date until 1930, he maintained a mining engineering practice with headquarters in Los Angeles, examining and managing mines in this country, Alaska, British Columbia, and Mexico. During most of this period he was a partner in the firm of Sill & Sill, which operated a testing and metallurgical laboratory in Los Angeles.

For the next 12 years, until 1942, he carried on a consulting practice, largely on heavy construction. He served on the consulting board for the Los Angeles Dept. of Water & Power on the Mono Basin project, in connection with the 11-mile Mono Crater Basin Tunnel, Bouquet and Lake Crowley Dams, and other masonry and earthfill dams of the department. He also served during this period as consultant to the Los Angeles Flood Control District on the construction of the dams in the San Gabriel Canyon in Los Angeles County, and as a consultant on a number of large earthfill dams and tunnels in the Western section of the U. S.

In 1942, he, with Carl B. Wirsching and Donald M. Baker, formed the firm of Ruscardon Engineers, with offices in Los Angeles. Mr. Wirsching resigned from this firm to accept appointment as City Manager of Long Beach, Calif., in 1947. During this period, Sill served in a consulting capacity for many large Western contractors engaged in dam and tunnel construction. He was appointed by the U. S. Bureau of Reclamation as a member of the consulting board on the Tecolote Tunnel in Santa Barbara County, Calif. He was widely known as one of the leading authorities on tunnels in the West.

He was past president of the Los Angeles Section of AIME, a member of the American Society of Civil Engineers, and a registered civil engineer in the State of California.

He is survived by his wife, Mildred, his sister, Mrs. Tom Frank Williams of Ft. Smith, Ark., and his brother, Harley A. Sill of Los Angeles.

Stanley M. Walker (Member 1948) died June 17, 1954 in Nepal. He was technical advisor on mining to the Nepalese Government and had been since 1953 when he accepted transfer from the DMPA to go to Kathmandu. Mr. Walker was born in Denver in 1887. His early mining experience was gained with Tezuitlan Copper Co. in Pueblo, Mexico, and summers, while studying at Colorado School of Mines, he worked for various Colorado mining companies. From 1911 to 1912 Mr. Walker was field and construction engineer with Nevada-California Power Co., Los Angeles, and then superintendent of Madonna mine at Monarch, Colo.; general manager of Monte Cristo mine, Ouray, Colo.; and superintendent, Frisco Tunnel Co., Animas Forks, Colo. After a period of consulting and petroleum geology work, Mr. Walker was general manager, Ward Big 5 Gold Mining Co., Ward, Colo. From 1939 to 1942, he was assistant director, Denver Housing Authority, and in 1942 he became senior mining engineer, U.S. Bureau of Mines, Birmingham. Mr. Walker was made acting chief, Housing & Utilities Branch, Requirements Div., DMPA in 1951.

Necrology

Date Elected	Name	Date of Death
1934	C. Augustus Carlow	Aug. 13, 1954
1949	E. Y. Ellis	Aug. 24, 1954
1926	Thomas W. Guy	Sept. 11, 1954
1937	C. H. Heist	Unknown
1928	Walter L. Kohlberg	July 23, 1949
1907	John H. Leavell	Sept. 15, 1954
1890	Spencer Miller	June 1953
1919	John Hunter Nead	Sept. 13, 1954
1954	Charles M. Paul	Sept. 1, 1954
1937	Roger H. Sherman	Sept. 8, 1954
1944	Rush T. Sill	Aug. 19, 1954
1951	Charles E. Sumpter	Sept. 13, 1954
1916	Arthur D. Terrell	February 1954
1948	Stanley M. Walker	June 17, 1954
1923	Nicholas A. Ziegler	Aug. 20, 1954

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Total AIME membership on Sept. 30, 1954 was 21,466; in addition 1738 Student Associates were enrolled.

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In the following list C/S means change of status; R, reinstatement; M, Member; J, Junior Member; A, Associate Member; S, Student Associate.

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Grand Junction—Simpson, Julian E. (M)

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New York
Manhasset—Dufour, Maurice F. (M)
New York—Revel, Humbert S. (A)

North Carolina
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Ohio
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Painesville—Richner, Donald R. (R. C/S—J-M)

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Portland—Watters, William H. (M)

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— Coming Events —

Nov. 15-17, Dallas Council on World Affairs, conference on U. S. raw material needs, Dallas.

Nov. 17-18, National Coal Assn., annual meeting, Hotel William Penn, Pittsburgh.

Nov. 17, Coal Exporters' Assn. of U. S. Inc., annual meeting, Hotel William Penn, Pittsburgh, in conjunction with NCA convention.

Nov. 18, AIME, Utah Local Section, joint meeting with Intermountain Assn. of Petroleum Geologists, 8:00 pm, Newhouse Hotel, Salt Lake City.

Nov. 18, American Mining Congress, Coal Div., Wm. Penn Hotel, Pittsburgh.

Nov. 18, AIME, Colorado Section, dinner, A. H. Shoemaker, general manager, Homestake Mining Co., speaker, 6:30 pm, University Club, Denver.

Nov. 18-19, Society of Exploration Geophysicists, 8th annual Midwestern meeting, Adolphus Hotel, Dallas. Registration Nov. 17.

Nov. 23, Manufacturing Chemists' Assn. Inc., 4th semiannual meeting and winter conference, Statler Hotel, New York.

Nov. 23, Society for Applied Spectroscopy, dinner, Lido Restaurant; meeting Philadelphia College of Pharmacy and Science, Philadelphia.

Nov. 28-Dec. 3, American Society of Mechanical Engineers, annual meeting, Hotel Statler, New York.

Dec. 1-3, AIME Electric Furnace Conference, William Penn Hotel, Pittsburgh.

Dec. 3-4, AIME, Columbia Section, joint session with Northwest Mining Assn.

Dec. 10, AIME, Lehigh Valley Section, annual dinner meeting (Ladies' Night), Hotel Bethlehem, Bethlehem, Pa.

Dec. 10, AIME, St. Louis Section, symposium on conveyors, Hotel York, St. Louis.

Dec. 12-15, American Institute of Chemical Engineers, annual meeting, Statler Hotel, New York.

Dec. 26-31, American Assn. for the Advancement of Science, national meeting, University of California, Berkeley, Calif.

Jan. 9-13, 1955, National Sand & Gravel Assn., 39th annual convention, Miami, Fla.

Jan. 12, AIME, Connecticut Local Section, Bridgeport, Conn.

Jan. 13, AIME, Cleveland Section, annual dinner meeting, 6:30 pm, Manger (Allerton) Hotel, Cleveland.

Jan. 17-18, National Agricultural Limestone Institute Inc., 10th annual convention, Hotel Statler, Washington, D. C. Executive committee, January 15; board meeting, January 16.

Feb 3-5, Colorado Mining Assn., annual meeting, Denver

Feb. 14-17, AIME, Annual Meeting, Conrad Hilton Hotel, Chicago.

Mar. 16, AIME, Connecticut Local Section, annual meeting, Statler Hotel, Hartford, Conn.

Mar. 28-Apr. 1, Ninth Western Metal Exposition, Pan-Pacific Auditorium, and Ninth Western Metal Congress, Ambassador Hotel, Los Angeles.

Apr. 18, Material Handling Institute, spring meeting 10:00 am, Drake Hotel, Chicago.

Apr. 18-19, Third National Air Pollution Symposium, Pasadena, Calif.

Apr. 19-21, Canadian Institute of Mining and Metallurgy, annual meeting, Royal York Hotel, Toronto.

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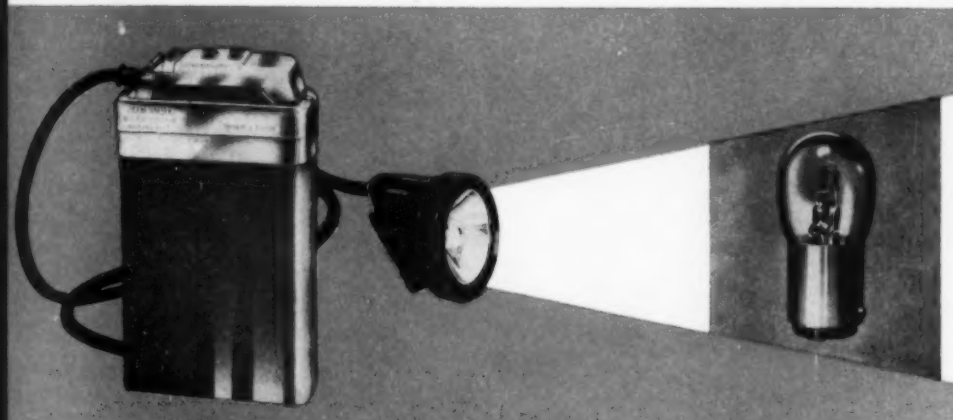

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